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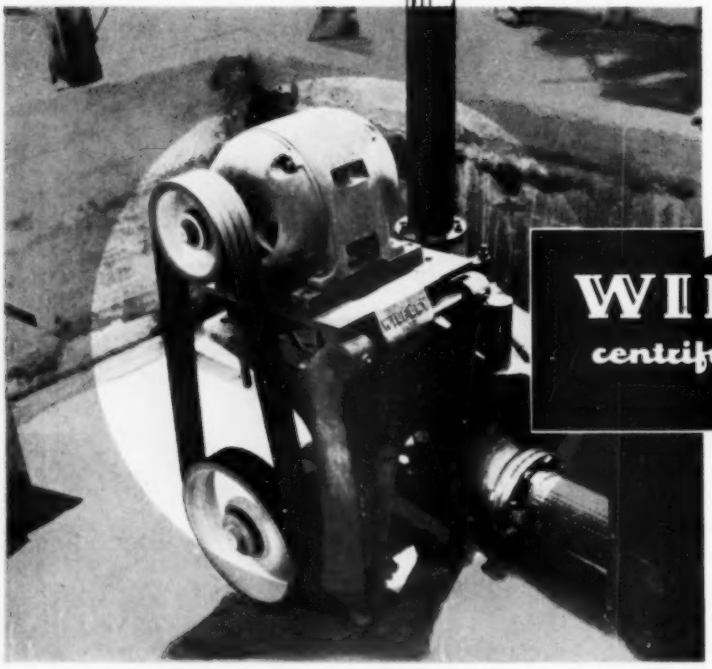

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VOL. 187 NO. 3 MARCH 1950

In This Issue

COVER

We hope our cover picture this month serves to carry with it some of the fine personality of the man—Donald H. McLaughlin, scholar, scientist and businessman, who will serve as President of AIME for 1950. The interesting story of his accomplishments appears on p. 406.

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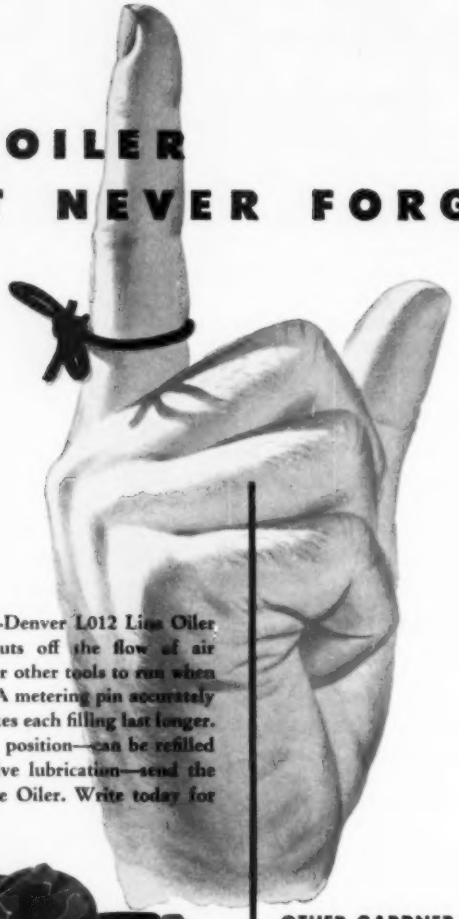

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Mining Engineer, 38, Dutch citizen, married, no children. Seven years' experience alluvial gold mining, tin, lead, copper metallurgy; three years manager Rockwool plant; two years operating consulting engineering office own surveying and field testing equipment, seeks position mining, construction, sales engineering. Interest Point 4. So. America, Africa or Far East development programs. Speak and write seven languages. M-514.

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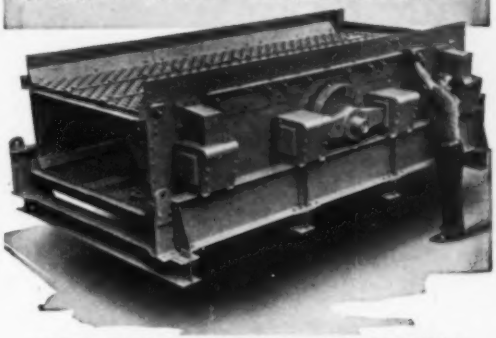

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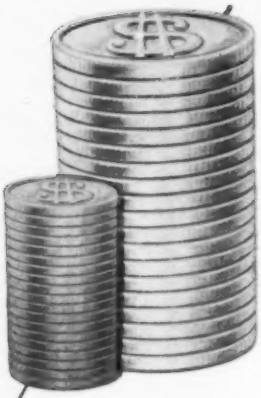

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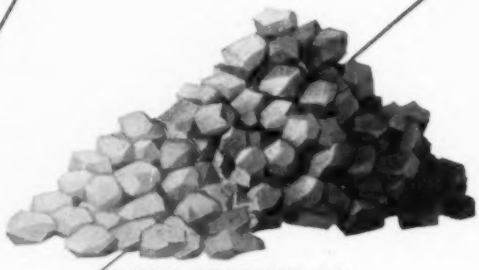

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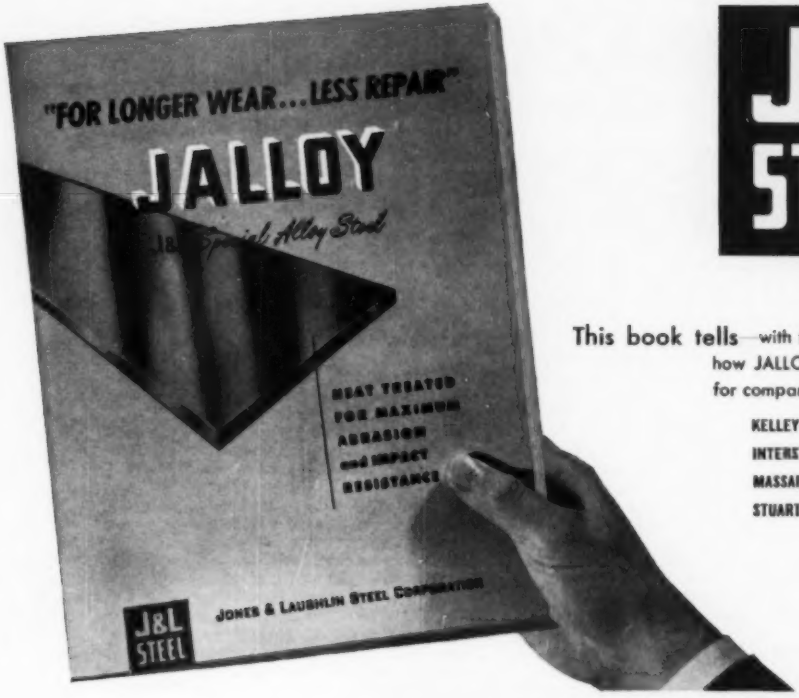
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Authors in This Issue

E. R. Kaiser

Elmer R. Kaiser (p. 374, although he admits a penchant for oil painting, still devotes the major part of his time and talent to the coal industry. Born in New Holstein, Wisconsin, Mr. Kaiser attended high school in Kiel, Wis., and then took B.S. and M.S. degrees, (cum laude) from the Univ. of Wisconsin. From 1935 to 1944 he was a fuel engineer with Battelle Memorial Institute, doubling as assistant to the president of Bituminous Coal Research, Inc. during five of those years. For the last six years he has been assistant to the Director of Research for BCR. An active AIME member, Mr. Kaiser is Program Chairman of the Coal Division, chairman of the Committee on Research, and is also chairman of the ASME fuels division. In addition to all this, he finds time to enjoy woodworking, metalworking, gardening, and the above mentioned oil painting. . . . **J. D. Morgan, Jr.** (p. 369), now assistant for materials and stockpile policies with the National Security Resources Board in Washington, D. C., was an officer with the Corps of Engineers, serving in the Pacific Theater during the recent war. Dr. Morgan attended the School of Mineral Industries at Penn. State College, taking his B.S., M.S., and Ph.D. degrees between 1942 and 1948. He also holds an E.M. degree from that school. He has coauthored two other AIME TP's, and has written for Mining Congress Journal, the Military Engineer, and other magazines. The NSRB recently published his Ph.D. dissertation "The Domestic Mining Industry of the U. S. in World War II.". . .

J. D. Morgan, Jr.

N. Ketzlach

Norman Ketzlach (p. 391) of Seattle, Wash., is chief chemist for Manganese Products, Inc., having held that position for the past four years. Prior to joining Manganese Products he had a teaching fellowship in industrial chemistry at the University of Washington, from which institution he holds both B.S. and M.S. degrees. While attending the University of Washington, he was elected to four honorary groups: Zeta Mu Tau, Tau Beta Pi, Phi Lambda Upsilon, and Sigma Xi. Mr. Ketzlach was born in Seattle, Wash., and attended Garfield High School there. In his spare time he enjoys photography. . . . **G. L. Simard** (p. 359), now a group leader with the research division of the American Cyanamid Co. in Stamford, Conn., was formerly a research engineer with the Battelle Memorial Institute in Columbus, Ohio. A native of Lewiston, Maine, he attended Bates College and MIT, taking his Ph.D. degree. He now resides in Redding, Conn. This is his first paper to be published by the AIME. . . **R. F. Wesner** (p. 369) holds B.S. and M.S. degrees in mineral preparation engineering from Penn State College, and has spent one year teaching at his Alma Mater. He has also served in the USAF for three years, and for the past year and a half has been a field service engineer with the McNally Pittsburgh Mfg. Co., Pittsburgh, Kansas. He became a student associate of AIME in 1942, and a member two years ago. . . . **J. Bernhardt** (p. 355) has been with nos. 3 and 4 mines of the Cornwall division, Bethlehem Cornwall Corp., Cornwall, Pa., for the last fourteen years, working his way up through successive positions of mining engineer, mine captain, and now functioning as mine superintendent. Mr. Bernhardt, an Institute member, attended the Univ. of Michigan and the Michigan College of Mining and Technology to gain his B.S. and E.M. degrees. Philately is his chief recreation. . . .

C. N. Savage

C. N. Savage, (p. 337) is an assistant professor of geology at Kent State Univ., Kent, Ohio. He took an M.S. from Northwestern in 1940, and for the past ten years has been engaged in teaching and public service, having worked on foundation studies for the Panama Canal, the New England Public Services, the OSS, and the Department of State. He has also taught at Colby and worked with the Maine Geological Survey. He was born in Fairfield, Maine, attended Lawrence H. S., and took his first degree from Colby in 1938. Prof. Savage has many hobbies, among them stamp collecting, wood carving, natural history, and photography . . . **J. J. Bean** (p. 365) is a mineral dressing engineer for American Cyanamid in Stamford, Conn., having been a field engineer with that firm, and having served in various capacities with Westinghouse, International Smelting and Refining, and the General Engineering Co. in Salt Lake City. He attended East High School in his native Salt Lake City, and has a B.S. in mechanical engineering from the Univ. of Utah. Although previously unpublished by AIME (of which he is a member), he has presented material before the ASME. Mr. Bean lives in Riverside, Conn., where he relaxes with his hobby—model railroading. . . .

J. J. Bean

Orville R. Lyons (p. 381) is another three-degree man, an Institute member, and has presented two other papers before the AIME. He lives in Pittsburgh, where he's a coal preparation engineer for Heyl and Patterson. He has been a coal preparation research engineer with Battelle Memorial Institute, and held a similar position with Republic Steel. Born in Staples, Minn., Mr. Lyons attended Central High School in Grand Forks, N. Dak., and then the Universities of North Dakota and Alabama, taking B.S. degrees in mining engineering and geology, and an M.S. in mining engineering. Mr. Lyons has been an AIME member for ten years. His hobbies include sketching, woodworking, stamp collecting, "and four children." **M. M. Fine** (see p. 21, December 1949.)

O. R. Lyons

S-Chuan Sun

Shiou-Chuan Sun (p. 369) is a prolific AIME writer, having presented a total of five papers since 1943. His three degrees and fifteen years' experience in the fields of mining and metallurgy doubtless explain why he writes so well, and so often. Born in Pung-Lai, Shantung, China, he attended Yu-Tê High School in Paoting, and then took his B.S. from Pei-Yang Univ. in Tientsin. Three years later he emerged from the Missouri School of Mines with an M.S. in mining. In 1945 he took an Sc.D. in metallurgy from MIT. During this time he worked as an assistant mining engineer in Shensie, China (1935-6), a research fellow with the U.S. Bureau of Mines (1945-6), a research engineer with MIT, and has been, for the past four years, an assistant professor of mineral preparation at Penn. State College. Mr. Sun has been an Institute member for thirteen years. When not teaching or writing TP's, he enjoys playing tennis, photography, and cooking Chinese food.

D. J. Salley (p. 359) was born in Red Lake Falls, Minn., attended high school there, and later took B.A. and Ph.D. degrees from the Univ. of North Dakota and Princeton Univ., respectively. He spent one year with the Bell Telephone Laboratories, and then, in 1932, became a Textile Foundation Fellow. Four years later, Dr. Salley went with American Cyanamid, and is now with that firm's research division in Stamford, Conn. . . . **J. R. Hoffert** (p. 340) is a civil engineer with a degree from Cornell Univ., who has spent the bulk of his career in Pennsylvania. He has served as assistant superintendent for the Harrisburg Park Commission, as a progress engineer on construction of Edgewood arsenal, and moved up through the ranks to become chief engineer of the Pennsylvania Department of Health. He is a member of ASCE and ASME, and has presented papers before other technical and engineering societies. Mr. Hoffert was born in Omaha, Nebraska, attended high school in Harrisburg, Pa., and at Cornell was awarded the Fuertes medal and elected to Sigma Xi and Tau Beta Pi. He has an "outdoorsman's interests," and enjoys shop work, gardening and photography.

Walter Hull Aldridge (p. 349) was born in Brooklyn, Sept. 8, 1867, and received his Engineer of Mines degree from Columbia Univ. twenty years later. In 1929, Columbia honored him with a Sc. D. degree and also its Egleston Medal. Upon graduation from Columbia, Dr. Aldridge went to Pueblo, Colo., to start his career. He has been president of the Texas Gulf Sulphur Co. since 1918; has been a member of AIME for 61 years, having been elected an Honorary Member in 1944; was awarded the AIME's W. L. Saunders Medal in 1933; and was the recipient of the John Fritz Medal for 1949. . . . **W. B. Plank** (p. 347), head of the department of mining and metallurgical engineering at Lafayette College, Easton, Pa., has written numerous papers for the AIME publications during his 32 years of membership. Born in Morgantown, Pa., in 1886, he received his B.S. in mining and metallurgical engineering at Penn State College in 1908 and his E.M. a year later. Before coming to Lafayette in 1920, he worked for the Philadelphia and Reading Coal & Iron Co., Pittsburgh Coal Co., and U. S. Bureau of Mines. Professor Plank has been very active in Institute affairs, having been on the Board of Directors for several years and secretary of the Mineral Industry Education Division. He is well known for his interest in, and numerous publications on, mineral industry education.

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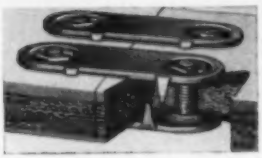

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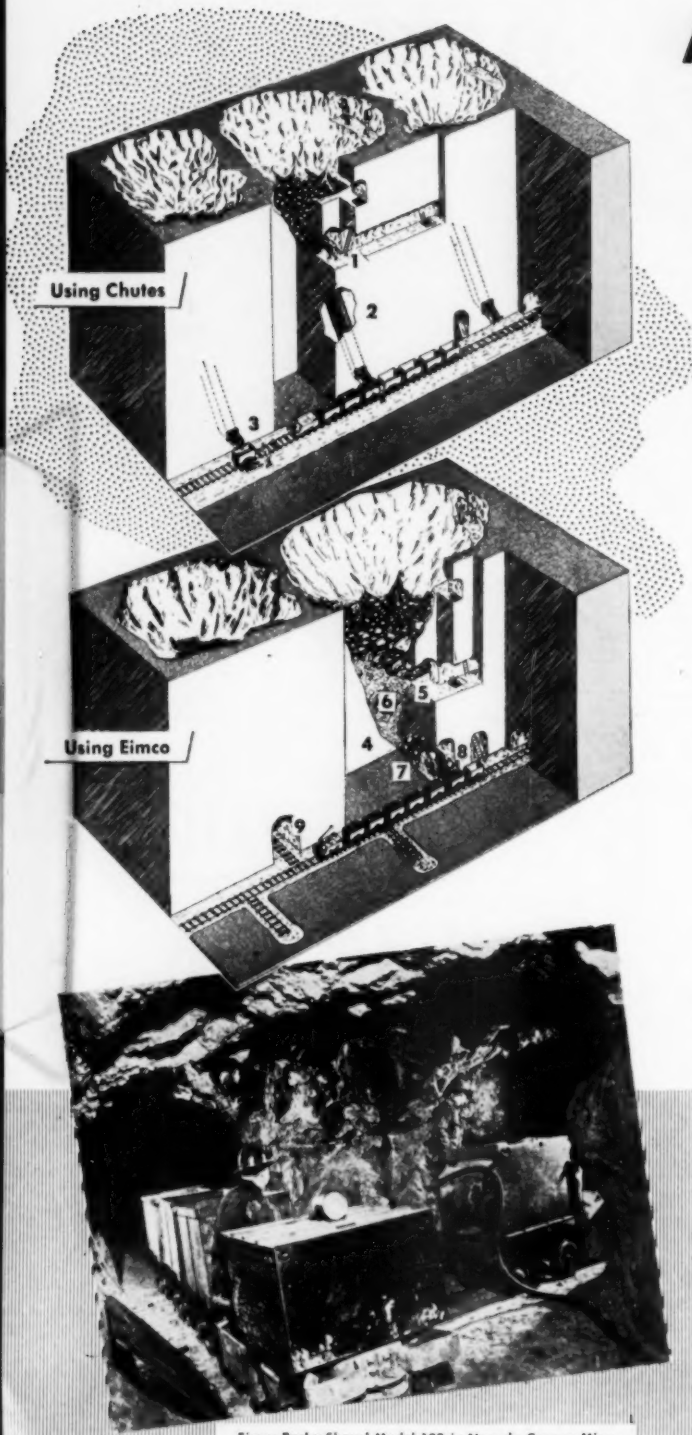

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MINING

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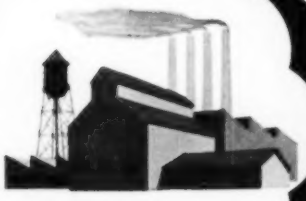

MANUFACTURING

MARINE

CONSTRUCTION

L. E. Bechman

Two 35-in. dia. by 30-ft Allis-Chalmers log washers.

How Log Washers May Mean Added Profits for You...

WELL-WASHED products meet higher cleanliness specifications, may cut costs of further processing.

The double rows of steel paddles you see in these Allis-Chalmers log washers effectively remove medium sticky clays from stone, sand and gravel, iron, zinc, bauxite, phosphates and other ores.

They revolve toward each other, produce a vigorous *scuffing* action. In addition, feed and wash water flow counter-current, producing a *scouring and sluicing* action. Result is a cleaning job that removes essentially all clay and dirt.

CHECK THESE FEATURES

- High carbon steel log paddles are secured by two bolts, easy to remove.
- Clear-water-lubricated rubber sleeve marine type bearings on lower end.
- Extra motor sheaves are provided to give different log shaft speeds.
- Cut tooth steel gears enclosed in oil-tight case result in increased gear life and safety for operator.
- Entire unit is rigid — drive frame assembly is welded to upper end of tank, mounted on common base.

Besides log washers, Allis-Chalmers builds a *complete line* of washing equipment for *any* application: Blade mills, for larger sizes of materials containing tenacious clays . . . sand screw washers, for washing and dewatering rock or ores in sizes ½-inch or smaller . . . revolving scrubbers and screens.

The Allis-Chalmers representative in your area may be able to show you how modern washing equipment can save you money. Call him, or write for log washer Bulletin 07B6411. A-2925

Texrope is an Allis-Chalmers trademark

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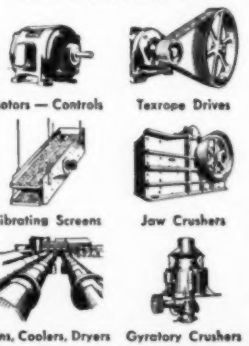

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The four cylinder piston-type reversible radial air motor equipped with safety type throttle supplies the extra power needed when the going gets tough. Wide band-type brake gives positive holding power and reversible motor permits lowering loads under power, while a convenient clutch lever located at the gear end of the hoist permits the drum to be disengaged for quickly pulling out cable.

Miners find the new BU ideal for moving equipment; hauling timber; handling shaker pans; spotting cars; hoisting drills, drill steel and other equipment between levels; hoisting and hauling pipe, etc. Call your Ingersoll-Rand branch office today for complete details and a convincing demonstration. Our Air Power Specialists are ready to show you how I-R Air Powered Equipment enables the miner to do more work with less effort.

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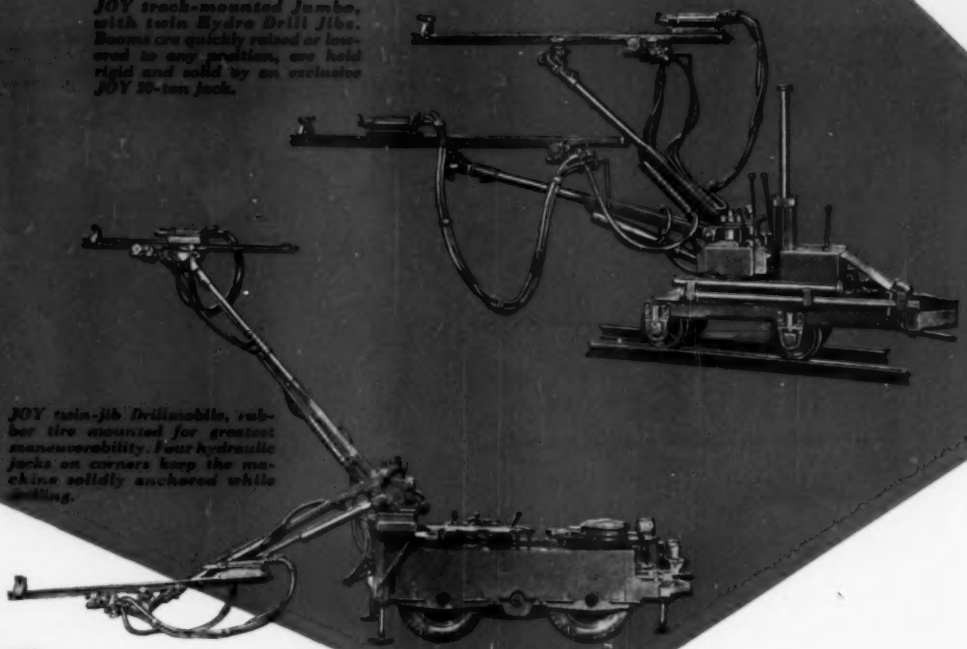

JOY track-mounted Jumbo, with twin Hydro Drill Jibs. Booms are quickly raised or lowered to any position, are held rigid and solid by an exclusive JOY 30-ton jack.

JOY twin-jib Drillmobile, rubber tire mounted for greatest maneuverability. Four hydraulic jacks on corners keep the machine solidly anchored while drilling.

Only JOY HYDRO DRILL JIBS

give you all these Advantages for FASTER ROUNDS *at* LESS COST

- ★ Long drill feeds—remote control of drills for easy one-man operation
- ★ Complete hydraulic control of jib positioning—pioneered by JOY
- ★ Drill holes more accurately spaced—correct angles more easily maintained
- ★ Greater speed in moving in—setting up—tearing down and moving out . . . no lost time
- ★ Drills holes anywhere in the face—no blind spots
- ★ Greater safety and no back-breaking labor for drill runners—they like it

ABOVE, Twin Hydro Drill Jibs with two JOY T-300 Drifters on a track-mounted Jumbo. Hydraulic control permits fast, easy positioning for any drilling angles. BELOW, a single Hydro Drill Jib model. Remote control and long feeds promote safety.

FOR BEST ALL-AROUND PERFORMANCE WITH JOY HYDRO DRILL JIBS . . .

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Nickel Alloy Steel Gears

There are two kinds of nickel alloy steel gears... those that are carburized, and those that are direct hardened.

CARBURIZED GEARS

The carburized gear is used in applications that require maximum wear resistance in the surface, as well as greatest surface compressive strength. With nickel alloy carburizing steels, this goal is consistently attained, together with development of extremely tough cores that resist shock loads, fatigue and bending stresses. Moreover, a chief cause of noisy gears... the distortion that accompanies heat treating... is inherently resisted by nickel alloy carburizing steels.

DIRECT HARDENED GEARS

The direct hardened steel gear is used to carry heavy tooth loading in applications where resistance to wear and surface compressive stresses is not quite so vital a factor. Here again, the nickel-containing steels develop the required strength more consistently and in heavier sections than carbon steels, and are generally more resistant to shock, fatigue and multi-axial stresses. Distortion resulting from heat treatment may be minimized by using nickel alloy steels and their machinability before final heat treatment is very good.

Giving greater play to the skill of the engineer, nickel alloyed steels not only provide increased strength without sacrificing ductility, but they harden at lower temperatures which simplifies heat treatment and minimizes deformation and scaling.

MEET VARIED REQUIREMENTS

Nickel alloyed steels enable producers to meet virtually any reasonable requirements . . . whether dictated by revised stress analysis due to design changes, or by changed fabricating methods that demand better machining qualities or improved response to heat treatment.

MANY TYPES AVAILABLE

The many standard grades of nickel alloyed steels permit specifying the particular type which provides the best set of properties for any reasonable fabrication and service demands.

Unending competition for higher speeds and heavier loads, for quieter operating and longer machine life, provide opportunities for gear producers to drive ahead with nickel alloyed steels. Use the coupon for your copy of "Modern Trends in Nickel Steel and Cast Iron Gear Materials." This useful and informative booklet is yours for the asking. Send for it now.

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EMBLEM OF INCO SERVICE
TRADE MARK

THE INTERNATIONAL NICKEL COMPANY, INC. 67 WALL STREET NEW YORK 5, N. Y.

* In 1949 the United States imported 7,400,000 tons of iron ore; Chile, Sweden and Canada, in that order of importance, supplied over 80 pct of this amount. U. S. imports have increased from 3 pct of the total supply in 1940 to 8 pct in 1949.

* Mica has been successfully synthesized by scientists of the National Bureau of Standards. Quartz, magnesite and bauxite, plus a fluorosilicate compound to act as a crystallizing agent, are heated to 2500°F in a platinum-lined crucible and as the furnace cools mica crystals grow. The key to parallel growth to make sheets is in precision control of temperature.

* Inland Steel Co. of Chicago has taken an option to lease from Steep Rock Iron Mines, Ltd., of Canada -- a large undeveloped acreage in the vicinity of Steep Rock Lake, Ontario -- for the purpose of exploring and developing a new iron ore mine. The exploration program is expected to begin at an early date.

* A 20-yr test conducted by the U. S. Bureau of Mines with the cooperation of Cleveland-Cliffs indicated that chemically treated timbers last from 2 to 3-1/2 times as long as untreated ones set in the same passageway. Of the three reagents used, zinc chloride gave the best result -- 13.3 yr as compared to 3.8 yr for untreated timber.

* Senator Pat McCarran charged that President Truman used a "shotgun approach" to the taxation problem when he accused the mining industry of enjoying excessive depletion exemptions when actually mining taxes should be lowered. He introduced to the Senate a report of the National Advisory Council recommending a decrease in mining taxes.

* The country's first graduate program of research and instruction in conservation of natural resources will begin at Yale in September. The Conservation Foundation is supplying $20,000 a year to finance the program.

* Felix E. Wormser urged that adequate tariff protection should be provided to offset effects of devaluation on the domestic mining industry in an address before the Colorado Mining Association. He advocates tariffs rather than subsidy legislation, such as S. 2105, for the protection of the lead-zinc miners because the latter would promote inefficiency and "impair our priceless free market."

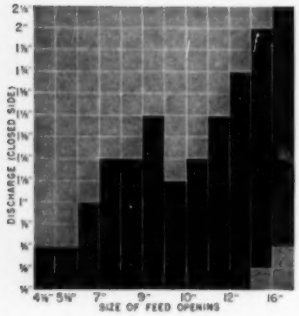
* "It may be startling to realize that for the world's white population the per capita requirements for minerals have multiplied 30 times in 30 yr, or that more minerals have been mined in the last 40 yr than in all prior history."
-- Evan Just, in "Mineral Depletion and Metal Supply."

* The National City Bank in its monthly letter states that both management and the unions concede that there are too many miners trying to make a living out of the industry for its present market.

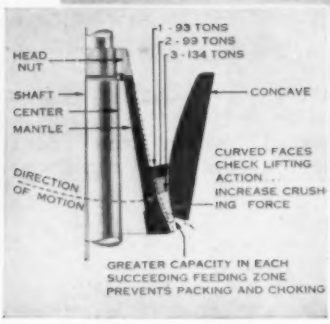
* On Feb. 3 it was reported that 6000 Anaconda employes in Chile quit jobs in sympathy with striking telephone and electric workers. The government has ordered them back to work on penalty of discharge. It appears that the walkout will not be prolonged in view of the government stand.

Beat Increasing Production Costs with Better Equipment

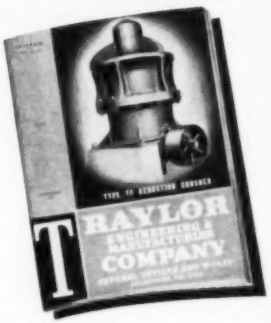
Increased plant efficiency is a logical means of boosting profits in the face of today's rising costs. A major factor in achieving this is mechanical excellence of your equipment. For reduction crushing operations, the Traylor TY Reduction Crusher is unexcelled for capacity and simplicity of design . . . trouble-free operation with minimum attention and maintenance.



The great working range of Traylor TY Reduction Crushers is shown by this chart. Each bar indicates the degree of adjustment of a Traylor TY Reduction Crusher *with a single set of head and concaves.* One Traylor TY gives you "job flexibility" greater than several less adjustable machines. This, with the TY'S great capacity and low power requirement, will increase your profits.



The non-choking, self-tightening bell head and curved concaves of the Traylor TY Reduction Crusher are clearly shown in this sectional view. These and the all around bottom discharge *with no diaphragm* are your guarantee of exceptional capacity and choke-free, pack-free operation. Patented dust seal and automatic, force-feed lubrication with a water cooled oil reservoir assure long service with minimum attention.



Bulletin 5112 shows how

Traylor TY Reduction Crushers . . . with their non-chokable bell heads and curved concaves . . . produce greater tonnages of a more uniform product remarkably free from slabs and fines. Booklet contains complete details and specifications. Write for your free copy today.

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A "TRAYLOR" LEADS TO GREATER PROFITS

Engineers Training Plan

EVERY forward-looking mining company realizes the need for employing graduate mining engineers so that there will be a supply of qualified men from which to draw for company officers, executives, and mine managers. The mining industry is competing today with numerous other fields of engineering for its supply of technical men, many of which are more attractive than mining because of better pay, living conditions, and well-regulated company training programs. The Ontario Mining Association and the mining companies of Ontario are conducting a postgraduate training plan which if adopted in the United States, might well encourage top quality men to study mining.

Under the Ontario plan, a selected group of graduates in mining and metallurgy may apply for one-year training. During this period, they are employed for four months at each of three mines—a small gold mine, a large gold mine, and a base metal mine. At each mine they work at all the jobs which the manager thinks would be of value to a young engineer, including activities underground, in the mine and the smelter, in the shops, the assay office, and the engineering and accounting departments. Payment is made by the company at the going rate for the job being done, but the graduate provides his own transportation to each mine. Neither party is obligated to continue the plan for the entire period.

A system of reporting on each trainee is used whereby the mine advises the Association, at the end of each month of training, of the various jobs at which the graduate has been employed. A four-month report is sent in, rating him on quality and quantity of work, adaptability, attitude, etc.; and comments on his suitability to the mining industry, attitude to safety, etc. The trainee is advised after the final report, in a general way, as to the ratings and suggestions made on the categories in which he should attempt to improve himself. Trainees completing at least six months of the plan receive a certificate showing that they have taken the course. Permanent jobs are not guaranteed but most trainees are hired by participating companies and the others have little difficulty getting placed.

There is no doubt that the companies incur expenses because of the plan, as does the trainee in moving from job to job, but the advantages for both parties so far outweigh the cost that the plan is being conducted for the third consecutive year, 1950.

We propose that the mining companies in coal, metals, and nonmetallics, in collaboration with the deans of the mining engineering schools in the United States, form a committee, possibly under the auspices of AIME, to inaugurate such a training plan. All companies should be encouraged to participate although they might not be able to accept a trainee every four months. Such a cooperative plan would permit the smaller companies to participate where it would be uneconomic for them to conduct their own company course. The larger companies could discontinue their own programs, if desirable, and draw on the trainees who would have broader training from working for more companies.

From both the standpoint of the mining company and the trainee, such a program would be profitable. Those who completed the course would have a broad, practical knowledge of most phases of the mining industry, enabling them to better orient themselves in their work and their job in the scheme of things. They would have worn off some of the restlessness of a recent graduate and inquisitiveness about other companies; questions on relative rates of advancement and the advantages of large versus small companies might be resolved. Mutual advantage in choosing a company or an employee would be derived based upon prolonged observation. Unity in the mining industry which is badly needed in this day of controversy on labor, subsidies, tariffs, and taxes, would be promoted by a cooperative effort of this nature.

We think that too much is to be gained by establishing such a program for the suggestion to be laid aside without comment. If those who are in a position to implement this plan give it some thought we feel sure that it can be speedily established, and will benefit all concerned.

It's Everyone's Business

FEB. 17—The past month found the average citizen backed off just a little more into his blind corner staring glassily at hydrogen bombs, unbalanced budgets, John L. Lewis, more inflation, a rising wave of strikes, the country's only active battleship on a mudbank, a famous scientist in gaol as a traitor, a favorite actress on a Roman holiday, and rising mountains of potatoes, butter and eggs begging for destruction.

About the only bright spot has been industrial activity and employment, but even these are menaced by the coal shortage. Most industries have orders on hand or in prospect to warrant high operations for some time to come. The trend of the Reserve Board's production index is upward except as strikes may interfere. There has been heavy buying at the seasonal merchandise shows, particularly in furniture and household goods. The demand for goods of this kind illustrates greater confidence in the Spring outlook than was generally felt a year ago. Further, the demand also reflects improved market positions due to curtailment of output and reduction of inventories in 1949 and sustained consumer buying power. It demonstrates the strong support that is given directly to many businesses, and indirectly to all, by the phenomenal construction activity that is now under way. The Commerce Department places the number of new dwelling units started in 1949 at the new all-time high of 1,019,000.

Automobile output has been cut some 30,000 to 35,000 cars a week by the strike against Chrysler. An offer has been made to provide employees $100 a month total pension and old-age insurance benefits upon retirement, which conforms to the pattern set in the steel and other cases. However, the union is striving to break old patterns and set new ones and the strike's duration is unpredictable.

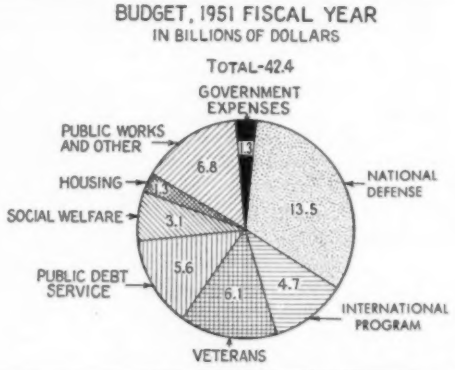
Steel producers are hard pressed to satisfy current demands. Among the base metals copper has held steadily at 18½¢ a lb; lead is unchanged at 12¢; zinc is steady at 9¾¢, and tin has slid off gradually to 74½¢ a lb. (See graph.) For the most part buyers of base metals are holding purchases to minimum requirements and this situation is likely to hold until the current strike wave moderates.

President Truman's budget for the fiscal year ending on June 30, 1951, much to the surprise of no one, showed the government again failing to balance its accounts even though peacetime prosperity has achieved heights which were hardly imagined until recently. A year ago the President estimated that the current year's deficit would be less than $900 million on what then seemed to be a rather shaky basis. But that deficit is now put at $5.5 billion. For the 1950-51 period the deficit estimate is only slightly lower—$5.1 billion—and if past experiences are any criteria the final figure will certainly be much larger. Even so, Mr. Truman obviously believes that the national economy is nicely balanced and that it should be possible to maintain stability and steady growth.

The 1951 budget (see graph) provides for an expenditure of $4.7 billion on foreign activities, $1.3 billion, or more than 20 pct less than is estimated for 1950. All segments of the European recovery program, including that portion of aid to western Germany which has previously been provided separately, will be more than $1 billion below the amounts provided for the same purposes in 1950. Only $35 million will be appropriated as the essential first step in the Fourth Point program for the development of backward areas.

With the cut in foreign items substantially greater than the decline in expenditure as a whole, the domestic items are allowed more in 1951 than in 1950. The difference is almost exactly accounted for by national defense expenditures which, including stockpiling and other ancillary items, are up, by $400 million, to $13.5 billion. Estimates for economic development have gone up, as has provision for welfare and social security while benefits to ex-servicemen are $825 million lower on the year.

In the prolonged coal dispute the underlying problem is always the over-expansion of the mines both in capital equipment and number of employees. Approximately two-thirds of the present industry, operated on a full-time basis, could supply all visible peacetime markets. With Mr. Lewis constantly dragging his feet as regards production, the supply problem would have boiled to a crisis much earlier except for the phenomenally mild winter, the steel strike, which eliminated one of the major sources of demand for coal and affected the requirements of related industries and, finally, the failure of the pension plan in its initial phase to reduce the number of miners as much as was expected.



The first sign of movement out of the coal impasse came with the filing by Mr. Denham, general counsel of the NLR, of a request for a court injunction against UMW for refusing to bargain in good faith and attempting to coerce the operators through the three-day week into renewing an illegal contract. The request for this injunction—which is not to be confused with the general injunction prohibiting for 80 days a strike that creates a "national emergency"—went to court, but six days earlier Mr. Lewis out-

maneuvered Mr. Denham by promptly accepting an offer of renewed negotiations from the operators, setting the very hour that Mr. Denham was scheduled to appear in court. The operators, however, postponed the meeting until after the court hearing.

The meetings sank beneath the quicksands of Mr. Lewis' usual histrionic mystery as regards specific demands and the operators' insistence on an unbreakable two-year contract and no increase in wages or pension levies. Mr. Truman then entered the arena with the proposal that all miners should return to a full working week for 70 days, while a fact-finding board investigated and drew up recommendations for an equitable settlement. Mr. Lewis' vitriolic reply to this proposal left Mr. Truman with no other recourse but to invoke the "Taft blackjack."

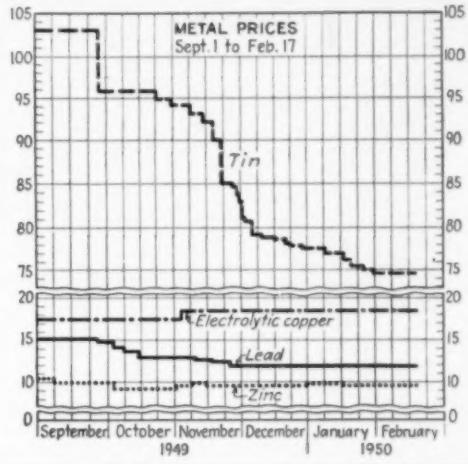
At this writing the miners still aren't digging coal and the next few days may see an injunction leveled at them. Even so, miners are a tough breed and the enforcement of the injunction will be an interesting operation to watch.

The hydrogen bomb, and all the attendant wide-eyed comment, came somewhat as a surprise to most scientists. It was the logical next step in the atomic arsenal. Its principle has been common knowledge for almost two decades awaiting only the atom bomb as a means of supplying the high temperature and pressure necessary to trigger the reaction. But while the next step forward did not surprise scientists, the action certainly was accompanied by a common feeling of revulsion. Atomic fission has at least the possibility of many beneficial peacetime byproducts. But the hydrogen version, atomic fusion, is purely a titanic instrument of destruction with no beneficial byproducts.

On the political front the hydrogen bomb has become primarily a device for scaring people into either a new arms race or a new attempt to come to terms with the Russians. The supporters of preventive war have spread reports that full-scale manufacture will soon begin, while those who think that the Baruch plan for international control of atomic energy is something less than sacred cite the hydrogen bomb as a reason for a more flexible approach to the problem in the UN. This is an election year and the Administration would like some dramatic success in foreign policy before November. Foreign aid programs have not the attraction they once had and the near defeat of the bill to aid Korea bodes ill for the forthcoming requests for money to further support European recovery.

With both the United States and Russia involved in a hydrogen bomb race, the state of world tension which has been normal for the past three years has been screwed up to almost an unbearable pitch. A new attempt to secure international control of atomic energy is certainly in the making. Mr. Churchill has spoken for another meeting with Mr. Stalin and President Truman, despite his denials, certainly is exploring a new attempt for agreement.

However, it is by no means certain that an agreement with the Soviet Union confined simply to the question of the atom bomb would either strengthen western defense or make the need for it less—the only two ways in which the risk of war could be reduced. In return for an American destruction of stockpiles and an undertaking not to continue with the hydrogen bomb, the Russians might agree to put into operation the system of international inspection and control of nationally produced atomc energy which is the maximum they have so far been prepared to concede. But would the risk of war be less even if the Russians made further advances and agreed to transfer some atomic activities to international ownership for an experimental period? On the contrary, the risk might be greater. Even a limited agreement with Russia might be taken as the signal for a slackening of vigilance and a weakening of the defensive effort all along the line. It is better that the cold war should continue until both sides are genuinely ready for a complete settlement than that any new scientific



horror should lead to the disarming of one side and the consequent tempting of the other to commit aggression.

According to the London Economist, even if no friendlier emotion ever stirs the Soviet leaders, surely the possibilities of the hydrogen bomb are horrifying enough to create a comradeship of commun funk. It would certainly be a grave error to allow this or any other opportunity of approaching Russia to slip. The hydrogen bomb does appear to offer a new opening. It should be seized, not simply to discuss the bomb, but to examine once again whether there is any possibility of a general *detente* between the east and west. Such a settlement could not be confined to atomic agreement; it would be dangerous if it were. It should cover all the major issues of policy—spheres of influence, the peace treaties, general disarmament, the elimination of hostile and inflammatory propaganda—which exacerbate tension and distrust in the world today. Only in such a setting of general pacification can specific agreements and concessions make a real contribution to peace. An atomic convention embedded in a larger framework of cordiality and restored confidence would give the world new hope. Without the wider accord there is a risk that it would only weaken the western powers, confuse their defense, divide their will and thus actually increase the likelihood of future war.

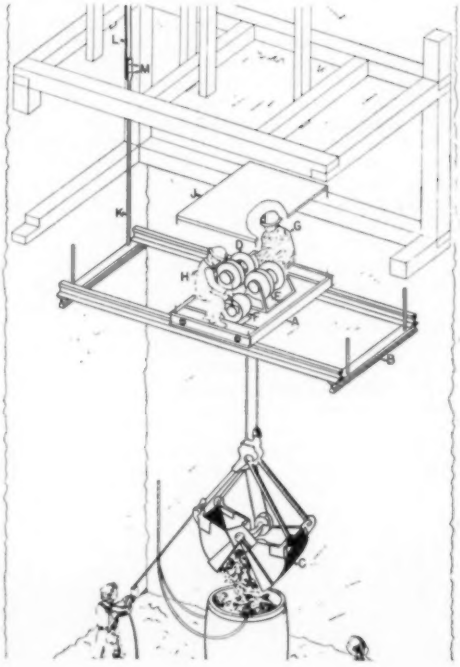
Machines for Efficient Shaft Mucking

- *Scraper and Slide*
- *Riddell Mucker*
- *Boskovich Device*
- *Hydro-mucker*
- *Inclined Shaft Mucking*
- *The Benching Method*
- *Performance Chart*

Hand mucking in shaft sinking operations is no longer necessary because several efficient methods and machines have been devised for this purpose for use in both vertical and inclined shafts. For years shaft sinking was considered the hardest and most dangerous job in mining and, consequently, the turnover in shaft crew was high. High turnover in personnel, particularly in shaft sinking, slows the operation because new men must be adjusted to the routine and peculiarities of the crew; accidents are more likely; and the cost of sinking increases. Mechanical shaft mucking has eliminated or alleviated the most disagreeable features of shaft sinking.

The purpose of this article is to give a brief sketch of the various methods of mechanical mucking; to give examples of performance data (see chart) with some discussion; and to provide a bibliography for reference.

The most important advantage of mechanized shaft mucking is the easing of the miner's toil. This single thing has made the entire sinking operation more efficient. The speed of the entire cycle depends largely on make-ready time, barring down, handling supplies and equipment, and timbering. These phases of the operation are made smooth by the cooperative effort of a well-organized team which is only achieved by long practice together. A. J. May, who is in charge of shaft sinking at the Star shaft for AS&R, credits the elimination of back-breaking mucking cycles, which contributed to the shaft crew turnover, with increased safety and efficiency. At the time of reporting, his crew went a year without a personnel change and racked up an advance of 290 ft for the 25 working days of May 1949. The Riddell system of mucking was used.



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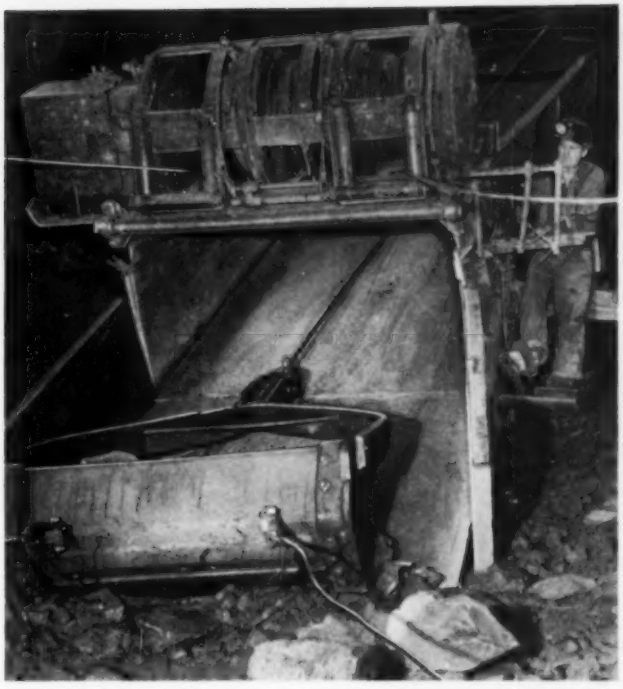
Hand shoveling is a thing of the past for clean-up in shaft sinking operations. Several methods have been developed for both vertical and inclined shafts, all of them reducing the time required to muck out and thereby realizing a saving in cost. If you are planning a shaft sinking job, you may well start your investigations for selecting a mechanical mucker by reading the following brief descriptions of the different machines and methods and studying the performance table.

• • •

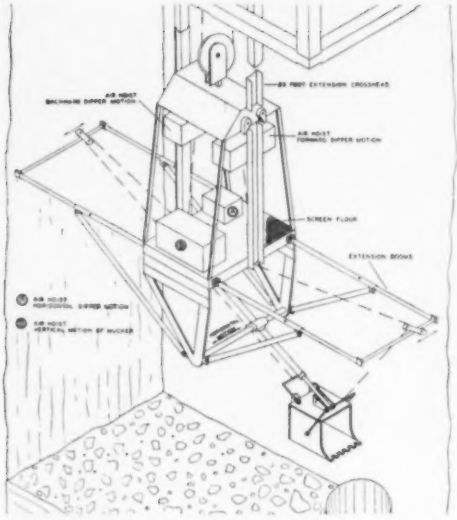


Scraper and Slide

In deepening its inclined Fisher Hill shaft in the Adirondacks, the Republic Steel Corp. mucked the broken rock by slushing with scraper and slide. Three tracks and a manway run the entire length of the shaft. The center and north tracks are used for hoisting ore in 20-ton skips in balance. The south track is for use as a man-and-material way, and also for handling some development muck. The center track is used for handling the shaft muck and is continually extended with temporary rails laid on the inclined rock floor of the shaft, so that it is kept within about 50 ft of the face. The scraper weighs 2500 lb, and the digging angle is 55 degrees to the direction of the pull. A Fisher Hill innovation involved use of a longer scraper slide, so that the inclined approach from the shaft bottom to the skip did not exceed 40 degrees. Scraping time was speeded 25% by the use of the longer scraper. Three shifts work with this scheme—one drills the round and fires it, one scales the roof, wets down, and moves the slide into position. The third removes the broken rock with the scraper and slide. The application of this longer slide and the sequence of operations made it possible to average a distance of 33.6 ft per week, or 7 ft per round.

Riddell Mucker

Conceived and developed in 1942, and designed to speed the expensive and onerous task of shaft mucking, the Riddell mucker is now in use in over twenty operations. In essence, it consists of a carriage, a frame, and a clamshell bucket. The steel carriage, on flanged wheels, moves horizontally along the long axis of the shaft, on a metal frame. Four hanging rods suspend the frame from the timbers overhead. Three air-driven motors are mounted on the carriage, one serving the digging line, one the hoisting line, and the third the carriage movement. Two men originally rode the carriage, one handling horizontal movement, and the other operating the clamshell bucket suspended from it. The use of a double-drum hoist has cut this to one operator. A light metal cover protects the men from falling water and solid objects. The frame on which the carriage rides is made of 112-lb heavy rail section, and is set within or beneath the timbered portion of the shaft. The clamshell loads directly into buckets, and no hand mucking is necessary. A 31-cu ft bucket can be loaded in 1½ to 2 minutes. Pumping may be unnecessary, as the clamshell bails the shaft bottom and drops both water and broken rock into the bucket. The single operator can achieve a high degree of control over the bucket, and can place it at any desired point in the shaft.



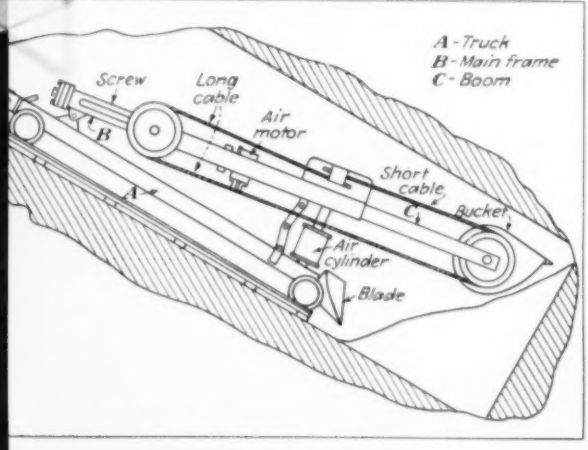
Hydro-mucker

Manufactured by the Hydrocrane Division of the Bucyrus-Erie Co., and used successfully at the Mather "B" shaft of the Cleveland-Cliffs Iron Co., the Hydro-mucker is a $\frac{3}{8}$ yd clamshell-type bucket weighing 1400 lb. A vertically mounted hydraulic ram actuated by 5000 psi oil pressure opens and closes the bucket. An IR-type hoist raises the bucket, it is swung manually over the muck pile, and lowered on the broken rock. When it descends, the ram closes the bucket, and it digs into the rock, coming up to fill a 2-yd tray which is then dumped into a mine car lowered into the shaft on the cage. The speed of mucking is dependent on the time required to dump the rock. Cleanup time at Cleveland-Cliffs was reduced 35 to 40% through use of the Hydro-mucker. A new twin-ram clamshell, with pistons mounted at 45 degree angles, is now being perfected. It is expected to outdo the original version, secure quick opening and better digging action.[4]

Boskovich Mucker

The Boskovich device, also constructed to operate inside timber, and easily placed in and removed from the shaft, was first developed and used at the New Park Mining Co's. Mayflower shaft in Keetley, Utah. This device features a power shovel dipper suspended beneath a mine cage. Four air hoists and an air piston provide swinging, crowding, and horizontal motion for the dipper. The cage to which the dipper is attached is constructed in accordance with the size of the shaft compartment, and its floor is a mesh screen, enabling the operator to see the shaft bottom. An "A" frame is attached to the cage bottom, and to this a length of shafting is fixed at right angles to the long dimension of the shaft. A machine clamp and then an air piston, both attached to the shafting, terminate in a $5\frac{1}{2}$-cu ft bucket to form the dipper for digging. The bucket is moved by cables, the cables passing through traveling sheaves anchored to extension booms which extend horizontally from the cage in the direction of the long dimension of the shaft. The booms are adjustable, consisting of $2\frac{1}{4}$-in. pipe inserted in 3-in. pipe. They can be quickly dropped below the cage when the mucker is hoisted for blasting. A man seated on the cage operates the dipper back and forth through the rock, using lever controls from two HU IR single-drum air hoists, and providing crowding action with a foot-operated valve controlling the aforementioned piston. In addition, three men work on the shaft bottom, leveling the muck pile, spotting buckets, and operating a sponge pump. Only one or two buckets had to be hand mucked per round, as the dipper proved highly efficient in the final cleanup.

At New Park, use of the machine doubled the speed of mucking shaft rounds, shortened the time required to timber, and eliminated the need for staging in drilling station operations. It was noted there that use of the Boskovich machine had a favorable psychological effect on the crew and also saved physical effort for other phases of the shaft sinking cycle. When used as constructed namely, inside timber, the hazard of falling slabs resulting from too much open ground is reduced.



Inclined Shaft Mucking Device

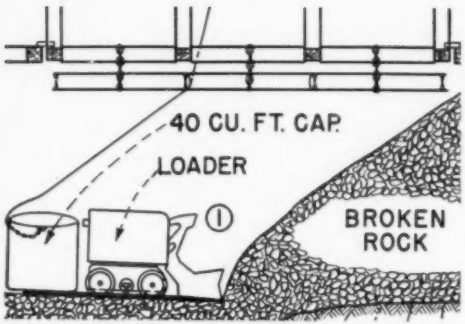
A self-loading skip good for 400-ft hauls has been developed at Bunker Hill for sinking an auxiliary shaft. Two machines are used, operating in balance in the shaft, which is sunk at a 50 degree angle. The frame is mounted on trucks. A steel section telescopes inside this frame, and the digging bucket is mounted at its lower end. When the machine is lowered, a miner at the shaft bottom turns on an air motor, which actuates a screw mechanism and retracts the extended telescoping section, to which the digging bucket is attached. The machine is then hoisted, the $\frac{1}{2}$-cu yd bucket dumping into a small pocket in the footwall of the shaft. The rock is then transferred by slushing to a waste pocket in the main shaft. One machine is being loaded as the other dumps. A Pacific digging lip is on the digging bucket, and a bulldozer blade on the main frame serves as a butt plate, and as a means of cleaning down the footwall of the shaft.

Performance Chart of Various Types of Mechanical Shaft Muckers

	Hecla Mining Co. Silver Summit Mining Co.	New Park Mining Co. Mayflower shaft	Quemont Mining Corp.	Falconbridge Nickel Mines, Ltd.	Cleveland-Cliffs Iron Co. Mather mine "B" shaft	Bunker Hill & Sullivan Mining & Concentrating Co.	Republic Steel Corp. Fisher Hill mine
1—Type of mucker:	Riddell Shaft Mucker	Boskovich	Model 21 EIMCO Rocker Shovel	Riddell Shaft Mucker	Hydro-mucker Hydraulically operated ⅝ cu yd clamshell	Own design	Portable scraper loading ramp
2—Feet sunk with mucker:	1562 ft	342 ft	1463.5 ft	437 ft to Aug. 1, 1949	1820 ft (from 1273 to 3093)	1100 ft	1500 ft
3—Angle of shaft:	Vertical	Vertical	Vertical	Vertical	Vertical	50 degrees from horizontal	33½ degrees
4—Type of Rock:	St. Regis Quartzite	Quartz Diorite Porphyry	Rhyolite Breccia—fractured	Norite	Jasper, Slate, and Quartzite	Quartzite	Granitic Gneiss
5—Cross sectional dimensions of shaft:	54 by 66 in. I.D.	9 by 19 ft	25 by 16 ft	6 ft 7½ in. by 19 ft 1 in. outside timber	13 ft 1 in. by 18 ft 7½ in. outside shaft steel	15 ft 2 in. by 7 ft 6 in.	10 ft high by 30 ft wide, rock section.
(a) No. of compartments:	3	3	5	3	4	3	4
(b) Description:		Two compartments 4 ft 6 in. by 5 ft 6 in., and one 5 ft by 5 ft 5 in.	Four compartments 6 by 6 ft, and 1 compartment 6 ft by 12 ft 9½ in. inside. Two skips, 2 cages, 1 manway.	Two hoisting compartments, 1 manway and ventilation compartment.	One cage compartment 12 ft by 6 ft 6 in., ladder and pipe compartment 12 ft by 3 ft 11½ in., two skip compartments 6 ft by 5 ft 8¾ in.	Two hoisting compartments, one pipe and ladderway.	Two 20-ton ore skips, 1 man cage, stairway and pipes.
6—GPM water pumped:	15 to 20 gpm	25 gpm	2½ Imp. gpm	2 gpm (six 32-cu ft buckets per 24 hr)	70 gpm	20 gpm	Minor flow pumped intermittently by air-operated sump pump.
7—What distance does shaft stand open below timber?	12 ft	20 to 30 ft	8 to 15 ft	18 to 20 ft	Ordinarily 20 to 40 ft	5 to 15 ft	Generally no timbering required except occasional fracture zone.
8—Nature of muck:	Medium	Medium with some coarse due to jointing	Medium	Medium	Varied from fine (in jasper) to coarse and flat in greywacke and slate.	Medium (some slabs)	Medium to coarse
9—No. of men in mucking crew at bottom of shaft, including machine operator:	4	4 per shift	8.6 men, average including hand mucking	4	9 men and boss (actually only 4 men needed but 9 makes for better all-around crew for drilling, setting steel).	4	6 men split 2 on each of 2 shifts, they scale roof and make ready for drilling.
10—Average time to muck out:	3½ to 4 hr	2 shifts with actual working time of 5 hr	8.7 work hr per bench	4 hr	13.4 hr, average	5 to 6 hr	About 8 hr
11—Average mucked each cycle:	55 cu yd	55 cu yd	81 cu yd	32 cu yd	149.6 cu yd	60 cu yd	About 120 cu yd
12—Average mucked, cu yd per man-hr:	3.7	2.75	1.08	2	1.2	2.5 to 3	7.5 yd per man-hr or approximately 200 cu ft of broken muck.
13—Percent of muck loaded mechanically each round:	100%	95% plus	70% per bench	100%	99%, machine rakes bottom	98%	100%
14—Initial cost of mucking machine:		Material $965.23 Labor 934.26 $1899.49 This was first machine built of this type and cost was therefore greater.	$5402	$6886.34	$2000	$5000	$7500 complete with hoist and scraper
15—Mucking cost per ft: Labor	$27.07	$29.02	$26.45 } 70% Mech. .63 }		$52.96	No record	Not available
Supplies		5.01 (other than machine)	$31.55 } 30% Hand .71 }		2.00		
(a) Former mucking cost per ft: (hand mucking) Labor			$105.11		$59.89	No record	
Supplies			2.41		Negligible This comparison based on 9-man crew maintained for better full cycle operation.		
16—Extent of delays due to mechanical failure of mucking machine:	Negligible	Negligible	Negligible	Negligible	Negligible	Negligible	Negligible
17—Increased rate of advance attributable to mechanical mucking:		0.12 ft per man-shift (0.28 compared to 0.16 ft per man-shift)	0.03 ft per man-shift	0	.03 ft per man-shift, again reflects large mucking crew.	75%	Figuring 20 cu ft per man-hr, the scraper will speed this part of the work ten times.
18—Hoist—double drum or single drum:	Double	Single	Double	Single	Single	Double	Three drum hoist, 50 hp
19—Capacity of muck bucket:	32 cu ft	21 cu ft	40 cu ft struck measure—average 29 cu ft net load	32 cu ft	54 cu ft	9 plus cu ft	150 cu ft
20—Hoisting capacity cu yd per hr or shift:							100
At start of sinking	16	15	14.1	8½	20	10	
depth	1530 ft	100 ft	100 ft	650 ft	0-1000 ft	600 ft	
At end of sinking	14	12	12.1	8	14	10	
depth	1592 ft	400 ft	1400 ft	1085 ft	1000-3000	1100 ft (new dump halfway down)	
21—Was time lost due to capacity of mucking machine being greater than hoisting capacity? If so, what percent of time was lost?	No	Yes—25%	Negligible	Yes—20%	Yes—30%	No	No

The Benching Method

Another method for eliminating hand mucking was used at the Quemont Mining Corp's. No. 2 (vertical) shaft at Noranda, Quebec. This idea involved the use of a Model 21 Eimco rocker shovel, and a benching method. Each half of the bottom of the long axis of the shaft was drilled, blasted, and mucked out alternately. The bench at one end was kept 2 to 3 ft below the previous bench, thus piling in one end the rock blasted at the other end. Benches were blasted out at an average depth of 5.5 ft, thus giving an average advance of 2.75 ft per bench. After blasting, some hand mucking is necessary to make room for the loader at the toe of the rock pile. A section of track with extension slide rails is lowered onto the long center axis of the shaft bottom, and the loader placed on the track. Six or seven men place two 40-cu ft buckets into position behind the loader. These buckets are hoisted in balance.



When the rocker has completed its work, hand mucking is again required to clean off the bench. While the loader is working, four to six men are completing the blocking of the shaft sets lining hoisting compartments, and placing guides and pipe.

Discussion

In looking over the performance chart (p. 335) care should be exercised in comparing data of one method with another because of such variables as character of broken rock, speed of hoisting, difference in cross sectional dimensions of shaft, difference in number of men in mucking crew for a balanced cycle of operations, etc.

All of the vertical shaft methods use the same procedure as for hand mucking to load the rock out of the shaft. That is, the rock is either loaded directly into buckets for hoisting or, as in the case of the Hydro-mucker, onto a tray which is dumped into a mine car on the cage. This means, of course, that an empty bucket must always be available to keep the loader busy and that the speed of cleanup is dependent on how fast the sinking hoist can empty the full buckets. Those operations which indicated that double-drum hoisting was used were able to keep the loading machine busy most continuously.

Item 12 gives the average cubic yardage mucked per man-hr; this figure varies from a low of 1.8 to a high of 3.7 for vertical shafts, and as high as 7.5 for the 33½ degree inclined Fisher Hill shaft. Taking into account that the Quemont low of 1.08 is attributable to 30% of the clean-up being by hand and that Cleveland-Cliffs only averaged 1.2 because of the large crew maintained for balance in the timbering and drilling phases of the cycle, it is evident that the machines are comparable and all speed the cleanup operation.

The Riddell mucker is reported to be able to handle large fragments impossible to move with a hand shovel. The Hydro-mucker is reported to have excellent digging characteristics because of the positive closing action of the hydraulic pistons. It has sufficient force to break rocks that tend to prevent the bucket from completely closing.

The inclined shafts show a marked increase in the rate of advance because of mechanical cleanup; this is less evident in the vertical shafts. In general, broken rock is easier to handle in inclined shafts, but in these two cases faster hoisting, and the mucking machines and methods should be credited. The Bunker-Hill design is particularly ingenious.

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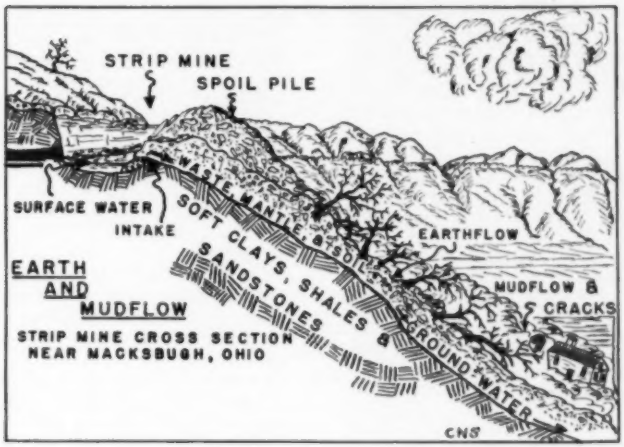
Earthflow



Associated with Strip Mining

The drawing above shows clearly the disastrous effects of a slow-motion "landslide," or mass wasting. A danger to water supplies, homes, and roads, it can also put strip miners out of business. Read how and why it occurs, and some suggestions for prevention.

by C. N. Savage

Mass wasting, or gravity-controlled movement of rock materials and mudflow, often creates major problems for construction and mining engineers. Various methods have been devised for dealing with such landslide-type phenomena (i.e., freezing, grouting, drying, etc.), but the more practical approach should be to forestall conditions that cause trouble. When factors favoring gravity-controlled movements can be recognized early, certain precautions may preclude complex and usually expensive complications. Such precautions frequently hinge on a practical approach to the job of excavating and constructing. This approach should deal with the possibility of mass wasting.

It is particularly important that the kind of rock materials and the water-bearing qualities of such materials be considered in light of the type of structure and methods of construction. How will the proposed operations affect the materials and their water-bearing qualities? Water plays two important parts in the role of an aid to mass wasting or gravity movement: (1) as a lubricant, and (2) as an increased load on unsafe bearing surfaces, i.e., a slope of high angle.

An examination of subsurface materials and determination of the bearing strength of these materials is usually one of the preliminary steps on most construction projects. This is done with a view to strength of materials—sometimes the problem of water-bearing characteristics is given too little attention. The normal relationship of the underlying materials to natural ground water —its accumulation and movement—is extremely important in foundation studies. Careful consideration of the possible results of excavation on conditions which already obtain is vital.

Fig. 1—Strip mine ditch on hillside intersects ground water table. Waste (left) is on downslope.

Fig. 2—Overburden thrown on downslope moves downward and upends trees and buildings.

Fig. 3—Earthflow and mudflow emerging from wooded area to damage houses at foot of hill.

Fig. 4—Earthflow has ensnared new house. In background, public highway is endangered.

In Ohio, controversy over the subject of backfilling of strip mines has waxed and waned. Many arguments have been set forth for the need of laws regulating backfill. Presumably, because of added costs and loss of time, the miners have protested such possible restrictions, the owners being responsible for such arguments. Strip mining for coal, however, has ramifications which go beyond the question of whether or not the stripped areas are unsightly or costly from the standpoint of future use of the land. There are other factors which indicate that it would be to the best interests of the operators to backfill the stripped areas. Excluding the possibility of deleterious effects upon local water supply, the operator of a strip mine, which is interfering with the movement of natural ground water, may leave himself open for trouble with mass wasting. This phenomena may cause loss of time and equipment by untimely covering of the operating face, or perhaps expensive lawsuits because of damage to the property of others.

In the spring of 1948, a particular example of mass wasting occurred, which offers much valuable data useful to anyone concerned with construction and excavation problems. The mass movement of materials as observed at Macksburgh, Ohio, has its genesis in factors which might well be present at any site where similar conditions prevail. These factors need not be peculiar to a strip mine, but may be present at other excavation sites.

Professor Savage is assistant professor of geology, Kent State University, Kent, Ohio.

Near Macksburgh, as a result of strip mining for coal, many cubic yards of earth and mud moved downslope, wrecking two sets of buildings, overriding agricultural land, destroying a wooded plot, and disrupting a through highway (fig. 2, 3, and 4). It has been reported that the strip mine operator has been subject to court action as a result of the damage.

The strip mining activities follow the contour of hill slopes fairly well inasmuch as the coal beds are nearly horizontal (fig. 1). At Macksburgh, power shovels are stripping near the top of the slope in a region of mature topography, with some relief. The coals are interbedded with sandstone and clayey to sandy shales; the shales are soft and easily broken up to form silt and clay. The cover or overburden above the coal is approximately 20 to 30 ft thick and is shoveled out to the downslope side. This procedure forms a broad, ditch-like catchment area which intersects surface runoff and the ground water table (fig. 1).

The ground and surface waters which are thus diverted move downgrade through the pile of waste rock (stripped overburden) carrying silt and clay as it seeps and percolates through the materials. This subsurface runoff adds weight to the waste rock which is already heaped at an unnatural angle on the underlying soil and mantle. The latter in turn lies on a clay shale bedrock surface. The weight of the water increases the force-bearing tangent to the slope, water and clay lubricate the materials, and

gravity movement begins. This movement, involving many cubic feet of earth and rock, is more nearly an intermittent flow and is strictly defined as earth and mud flows, not landslide.

At Macksburgh, the effects of loading and lubrication have resulted in movement of soil and mantle downslope and trees thrown askew at strange angles (fig. 2 and 3). The sward and mantle have moved and resemble a great wrinkled carpet. Irregular ridges alternate with cracks and fractures where the sward has pulled apart under tension. In places, mudflows consisting of water, silt, and clay have oozed out the cracks and mud pools have formed (fig. 5). Further up the slope the waste overburden is tumbling and moving downslope (fig. 2). Movement is probably slow and intermittent, but relentlessly downward under the influence of gravity.

This sort of earthflow and mudflow has doubtless been recognized by many. Particular conditions of cause and effect may be slightly different, but the phenomenon has occurred frequently and will occur again in many cases. Reports indicate that such movement of earth materials is not uncommon in other parts of Ohio where strip mining is underway. The solution to this problem, created by poor stripping technique, probably lies in backfilling as the mining proceeds. It would seem that suitable methods for backfilling could be devised without too much trouble or expense. Perhaps machinery needs to be designed which will answer the purpose. Such replacement of materials would also help alleviate the common complaint concerning the damage to the landscape by stripping methods. If so, and if the filling is performed soon enough, the earthflow and mudflow will not present the same potential hazard that they do now at many operations. Suitable precautions and provisions for proper drainage should help take care of the situation when such mass wasting problems arise in connection with other types of excavation. The important thing is to recognize the danger of gravity-controlled mass movement, and then do something about ground water, subsurface drainage, and improper loading of bearing surfaces before serious complications develop.

Fig. 5—Mudflows of water, silt and clay emerge from cracks in the earth to form mud pools. Silt and clay carried by ground and surface waters continually "percolate" through stripped overburden.

Books for Engineers

A.S.T.M. Standards on Coal and Coke, prepared by A.S.T.M. Committee D-5 on Coal and Coke, October, 1949. American Society for Testing Materials, Philadelphia, 729 pp., illus., diagrs., charts, tables. $2.00. This booklet brings together all of the A.S.T.M. standards on coke and coal. It gives 28 test methods, specifications, definitions of terms, and numerous proposed methods. Coal sampling, analysis for volatile matter in connection with smoke ordinances, grindability, drop shatter test, tumbler test, screen analysis, size, sieve analysis, cubic foot weight, index of dustiness, and free-swelling are covered for coal and coke.

Tin, its Mining, Production, Technology and Applications. (ACS Monograph 51). By C. L. Mantell. 2 ed. Reinhold Publishing Corporation, New York, 1949. 573 pp., illus., diagrs., charts, maps, tables. $10. — A comprehensive and thoroughly critical monograph which assembles all the authoritative available information for the use of those in the various fields in which tin is used. This second edition discusses new practices and procedures developed in ores and ore deposits, mining and ore dressing, smelting and metallurgy, tin conservation, tin uses, corrosion and analytical methods. Of special interest is the use of secondary tin including German, Continental, and American practices.

Earth Waves—John Wiley & Sons has announced the publication in January of Monograph #2 in the Harvard Monographs in Applied Science, a new series of high-level books reporting the results of university research published jointly by the Harvard University Press and Wiley.

Entitled "Earth Waves," this second monograph is the work of L. Don Leet, professor of geology at Harvard and seismologist in charge of the Harvard Seismograph Station. A compact summary of present knowledge of waves of the earth, the monograph discusses their application and use in prospecting for minerals, studying hurricanes, measuring vibrations from dynamite blasts to determine their effects on buildings, and mapping broad earth structures of geologic interest. Results of recent research on types of wave motion, interpretations of refraction profiles, and microseisms are also presented.

Magazine changes name—The magazine *"Archiv für Metallkunde,"* published in Weinheim/Bergstr., Germany, has changed its name, effective Jan. 1, 1950, to *"Werkstoff und Korrosion."* It will now include the nonmetallic mineral field, having acquired Dr. Erich Rabald of Mannheim as co-editor.

Books may be purchased through I. P. Klein, Book Department, AIME. Whenever possible 10% discount is given. Order government publications direct from bureau concerned.

Abating Stream Pollution

. . . in the Anthracite Coal Fields

by J. R. HOFFERT

ON Oct. 27, 1941, the Sanitary Water Board of the Commonwealth of Pennsylvania called a meeting of the representatives of the coal operators in the Schuylkill River Basin, and frankly suggested to the operators that they "clean up their house" before someone else came in and did it for them; that they should not emulate the railroads with their "public be damned" policy and its fateful results.

It had been four years since the passage of the antipollution law of 1937, a law which exempted silt from collieries until such time as a practical method existed for the removal of their water polluting properties. It had been fifteen long years since Sisler, Frazer and Ashmead concluded that 900 million tons of culm and silt had been deposited in Pennsylvania's streams in the process of preparing five and a half billion tons of anthracite for the world's consumers. Fine coal was being "mined" in the streams by "washeries" and river dredges downstream from the collieries, and silt banks no longer held back the endless black stream entering the Schuylkill at the rate of 350,000 tons per month. The coal operators were apparently working on the outmoded assumption that each ton of fine coal prepared displaced a ton of larger sized, more profitable coal. The meeting between the Sanitary Water Board and the coal operators signaled the beginning of the end of Pennsylvania's pollution problems.

As a result a committee was formed at that meeting by the operators to find out what conditions were at each colliery. The Bureau assisted in the indication of methods of measurement and sampling and in the development of forms for reporting the needed data. One large producer had its chief engineer supervise the gathering of the information concerning the quantity of waste waters, the amount and sieve analyses of the waste solids, and the existing means for retaining

Froth flotation systems are recovering profitable coal once sent downstream. This is a 180 ft. diameter Dorr clarifier producing well clarified water at the Coaldale breaker of the Lehigh Navigation Coal Co. in Tamaqua.

PENNSYLVANIA'S Schuylkill River was, in November 1947, carrying a silt burden of 350,000 tons per month. By March of last year an eight-year program of State research and fine cooperation by the operators had reduced this figure to 7,000 tons.

them. The Committee's report of August 10, 1942 indicated that whereas it was estimated that in 1920, 1,800,000 tons of silt had been discharged to the streams in the Schuylkill Basin, in 1940, the latest full year for which statistics were available, only 528,000 tons reached the streams from the "legitimate" operators who, however, produced just under 62% of their 1920 output. On a pro rata basis this meant keeping 948,000 tons of silt out of the Schuylkill and its tributaries during 1940.

We investigated some 30 collieries on a flying trip in the summer of 1943, located in all four anthracite fields and ranging from the small to the largest in the world. From these the Bureau selected 25 collieries which would be representative of the various kinds, specific gravities and fracturing characteristics of anthracite coals, the various methods of coal preparation, sizes of collieries, and sources of coal, such as deep mine, stripping, and bank coal. Of utmost importance were conditions whereby quantities of flow of waste waters could be satisfactorily measured with suitable hydraulic conditions for weir settings where necessary, with freedom from upstream silting, and satisfactory opportunities for sampling.

Mr. Hoffert is chief engineer, department of health, Commonwealth of Pennsylvania. **This paper was presented at the fall meeting, Pennsylvania Anthracite Section, AIME, Oct. 1949.**

With the cooperation of the operators, a test was made by a party of engineers of the Bureau at each of the collieries selected, this test extending over a full day's operation. The flow of waste waters was measured and sampled every 15 minutes in such a way as to get a representative sample.

The composited samples collected were then sent into the Bureau's laboratory in Harrisburg and put through a series of tests which showed not only the total quantity of solids being discharged to the stream during the day of the test, but also the weight per ton of marketable coal, the readiness with which the waste solids settled, and how the collieries compared with each other. We also ran settling tests in the laboratory with carefully sized tiny silica spheres and with various sizes of the colliery waste solids themselves in order to compare these results with Stoke's Law of sedimentation.

A meeting of representatives of the coal operators from all four anthracite coal fields was then held at Harrisburg in January of 1944.

As a result of this meeting the original Schuylkill Basin Committee of four was expanded to include ten operators representing all four anthracite fields, with Evan Evans of the Lehigh Navigation Coal Co. as chairman. Attention to desilting of colliery waste waters was thus extended to all four fields. The time had now come to consider removal of the exemption accorded to silt under the antipollution law. Accordingly the Bureau drew up a proposed resolution to be adopted by the Sanitary Water Board for removing this exemption, as the Board was empowered by the act to do, and setting forth the degree to which the silt was to be removed. We had, of course, first secured an opinion from the Department of Justice concerning the Board's legal right to decree a removal of the exemption to such a degree as the Board might specify. Some of you will recall the meeting of the representatives of the anthracite industry and the author at Lansford on May 22, 1944 at which the proposed engineering criteria were discussed before their adoption by the Board, and the kind remarks which were made concerning the fairness of the proposed criteria for controlling the removal of silt from colliery waste waters.

The Board had previously decided that it would remove the exemption in one resolution and would set forth its requirements in another action, and at its meeting on May 17, 1944 had removed silt's long-standing exemption.

Thus was set in motion on June 7, 1944, the Sanitary Water Board's compulsory program for keeping colliery silt from the waters of the Commonwealth. Nearly a year later the amendment to the antipollution act, the "Brunner Bill," Act No. 177 of May 8, 1945, wrote into statutory form what the Board had already done by its authorized resolution.

Results of the Program

The operators had to be convinced that the Board really meant business, and then they had to gather the data for the reports upon their wastes. We prepared directions for measuring, sampling and testing their wastes and generally assisted them. In addition some of the technical men of the larger operators helped the less experienced operators and the data upon the actual conditions at the various collieries began to accumulate.

We constructed a four-inch-diameter lucite tube, familiarly known as "Lucy," 8 ft 9 in. long in two sections with ⅛-in. sampling petcocks at every foot in height. This tube was set in a cradle so constructed that after the tube had been filled to the 8-ft level with a sample of colliery waste water, it could be capped with a lucite disk, swung to the horizontal and rocked up and down to insure uniform dispersion of the solids. It was then swung to the vertical and locked in position while a two-hour sedimentation test was run, with samples collected at appropriate intervals through the petcocks for analysis for suspended solids and turbidity. We first used this tube at the Locust Coal Co. at Shenandoah Oct. 4, 1944 and it has been in use ever since.

As an example, Edgerley and Edwards, after testing the wastes at one colliery, estimated the life of its desilting basin at 228 days despite the superintendent's insistence that it was good for at least three years. Two hundred and twenty days later they again inspected conditions, found the silt pond close to overflowing and the colliery forces hurriedly raising the dikes. Of course, this "on the nose" check on their estimate was unusually close but the tube did prove to give such a close evalution of the solids content of the waste waters that it saved waiting for the results of the more refined laboratory tests.

Despite our various tests we were reluctant to specify a standard of a given number of parts per million of solids as the allowable amount until we had determined what was genuinely practical on the basis of full-scale tests with many types of coal and with due regard to the build-up of solids during the day in mechanical thickeners through recirculation. We told the operators that silt basins with effectively designed inlet and outlet arrangements, giving a nominal detention of not less than three hours and with a specified pool depth and freeboard, and mechanical clarifiers with an upward velocity not exceeding 3 ft per hr would, except in unusual cases, meet the Board's requirements. We have not had to revise our ideas and as a result of our experience we have held to a standard of not over 1000 ppm of suspended solids in the water discharged to the streams. Actually we have quite generally bettered this figure and in many instances the collieries are discharging to the streams water which resembles spring water in appearance.

The operators next had the problem of preparing plans for their desilting devices. This was a novel procedure for many of them and considerable time and some education was required before acceptable plans were finally received. These plans were as varied as the collieries for which they were made and they ranged from simple, small lagoons, or settling basins, to complicated froth flotation plants.

However, the settling basins did have several things in common. We insisted upon a pool of quiescent water of adequate area and not less than 24 in. in depth, proper locations of inlets and outlets; tight outlet stacks of adequate structural strength, with level weirs of a length to insure low approach velocities to minimize outflow of fine solids, a freeboard of at least 18 in. and structurally sound dikes. A few of the basins had peripheral inlet channels and central outlets, which have the advantage of producing a low head against the sides of the basin, and a saucer-shaped cross section. Appropriate requirements were made for the other types of desilting devices. We also required adequate reserve capacities and duplicate equipment, or spare parts where failure of such equipment would result in discharge of silt to the streams. Interlocking of the main breaker pumps with the waste water pumps and sumps with sufficient capacity to care for the water in the breaker waste-water system is a standard stipulation to prevent overflow.

I would like here to pay tribute to the cooperation which we have received from the coal industry. We did, of course, encounter a few cases in which individual operators had to learn that we were not interested in who they knew but only in how they were meeting the Board's requirements. Let us see what have been the results of this cooperation as shown by the statistics of silt kept from the streams.

Of the 43 wet-process collieries now operating in the Schuylkill River Basin all have desilting works in operation. Whereas even as late as Jan. 1, 1946 they were discharging an estimated 4600 tons a day of silt to the streams, this has been reduced to a mere 29 tons a day. In November 1947 the Schuylkill was carrying a silt burden of 350,000 tons a month; in March 1949 it carried 7000 tons and this, of course, was largely due to erosion from old banks. Including what the collieries were formerly keeping from the streams, it is estimated that 2,000,000 tons of silt a year are being retained on the land.

The Swatara is now freed from silt discharges since all fourteen collieries have desilting works which now permit only 5 tons a day to reach the stream instead of the 5235 tons reaching the creek on Jan. 1, 1946. Similarly Mahantango Creek, through six desilting works, has been freed from silt. Catawissa Creek, although it never received large quantities of silt, has been protected through four similar works. Black Shamokin Creek with nine desilting works completed, five under construction, and four more to go has had its silt discharges more than cut in half but still looks black due partly to the silt yet to be removed and partly due to textile dye. Mahanoy Creek, with nine completed installations, four under construction and one more to go has had its silt load cut to a third of that on Jan. 1, 1946, although to the inexperienced eye, the water in the stream appears to have undergone little improvement.

Seven completed installations and one under construction have completely stopped the discharge of silt to the Lehigh River. Of the 31 collieries in the Lackawanna River Basin, 21 have completed desilting works, two are under construction, and four are not operating. The net result of this has been an estimated reduction in

Lack of space was a problem to be solved by the Operator's ingenuity. In a congested valley area, one company built this hydroseparator atop third story of a building housing froth flotation equipment. In the left background there are two batteries of disc filters for dewatering fine coal.

silt discharge from 3750 tons a day on Jan. 1, 1946 to 565 tons a day at present.

In the basin of the North Branch of the Susquehanna there are 31 collieries of which seventeen have completed silt control works, four have them under construction, and seven are not operating. This has meant an estimated reduction from 4890 tons a day on Jan. 1, 1946 to 1900 tons today.

We believe these figures speak for themselves. The work is not completed but it has progressed so far that the river washeries and dredges have already been seriously affected and they have recognized the "handwriting on the wall." Small wonder when the reduction in silt discharges in the four fields is estimated at 15,200 tons a day.

The most satisfactory result of the silt control program in addition to the restoration of the streams is the recovery of the vast quantities of good Btu's, which formerly were sent down the streams. This has brought a substantial return to many of the collieries. Of course there have been problems, but a considerable body of experience has already been developed and the headaches are gradually being cured. One of the largest producers in the lower fields is about to install two froth flotation plants in addition to the three such plants already in operation by two other operators.

One of the biggest surprises for many of the operators was the rapidity with which their settling basins filled up when they were made real sedimentation devices instead of silt banks over which the waste waters traveled at relatively high velocity carrying away a large part of their suspended solids. This naturally requires a constant check on the operation of the desilting works. Treatment works are no better than their operation and we have a continuing "policing" job before us. We hope that it can be done in the same friendly way in which we and the anthracite industry have been cooperating thus, far.

The Board's program of stream improvement is being applied upon a state-wide and equitable basis against all forms of stream pollution, and hence the Board is ordering the preparation of plans and the construction of treatment works to correct stream pollution caused by the discharge of municipal sewage and objectionable industrial waste as well. The Board does not require this of streams impregnated with acid mine drainage because at the present time acid mine drainage is exempt under the provisions of the Anti-pollution Act for the reason that a practical method for removing the polluting properties is not known. However, the Sanitary Water Board has had two research projects underway for several years at Mellon Institute in an effort to find a reasonable and practical method for controlling or treating this drainage. A solution to the problem must be found. In pursuance of its abatement program the Board has issued orders to 559 municipalities, institutions and industries covering their sewage discharges, and 674 notices to industries, regarding their industrial wastes. More notices wil be issued as more investigations are completed.

This fifty thousand tons of small sized coal recovered at a downstream "washery" appeared formidable, but it represented only one-seventh of the amount entering the Schuylkill River each month, even as late as 1947.

Auxiliary Equipment Used in....

by Charles A. Lindberg

It will be generally agreed that the most important accessory to a rail haul operation is the railroad type locomotive crane. The oldest and still most common type is the steam crane, a self-contained steam power plant, coal fired, complete with boom and hoisting machinery and mechanical drive capable of propelling the unit on its own power at speeds up to 15 mph on the level, and hauling, if necessary, a car for transport of the objects being handled. Usually, for greater mobility and safety, this crane and flatcar are moved around the pit by a switcher locomotive. Crane capacities range from 25 to 40 tons, these loads being handled with the shortest radius of boom and with outrigger. Boom lengths are 55 to 60 ft long, usually with heavy lift main blocks at 40 ft boom lengths. Lifting capacities with booms at maximum radius are approximately 20% of maximum rating. Free capacities, namely without outriggers, range from 65 (at minimum radius) to 55% (at maximum radius) of the above capacity values.

Cranes are used extensively in the assembly and maintenance of shovel equipment, building of trestles for rail haul stripping dumps, laying, picking up and storing of pit track sections, loading, moving and placing heavy supplies, drill rigs, portable compressors, pumps, arc welders, etc. They are frequently called on for wrecker service, and with attachments can be used with clamshell buckets for digging and loading. They have proved very rugged, reliable, and long lived but are being superseded by the newer Diesel-powered railroad cranes which were first introduced on the Mesabi Range about three years ago. These new cranes are larger and heavier than the largest steam cranes and have maximum capacities up to 50 tons at 12 ft radius with outriggers. The booms are 55 ft to the point block and 50 ft to the main blocks. They can travel, when self-propelled, up to 15 mph and can be towed at speeds up to 40 mph. They also have important operating economies which will increase their use in the future. One weakness has been the limited capacity of the mechanical propulsion drives which have made it almost compulsory to move them with switcher locomotives when used in the pits.

The first Diesel cranes were delivered to the range within the past three months. They use the main Diesel engine to power the hoisting mechanisms with a mechanical drive, but propel with two 110-hp electric traction motors which are driven by a 230-v d-c generator direct driven by the 225-hp Diesel engine. These new cranes are of 55-ton maximum capacity, have 55 ft booms with the main block at 50 ft boom length, and are supplied with clamming attachments. Top traveling speed is 20 mph when self-propelled. It is a little early to evaluate the advantages of the new propulsion drive, but no serious troubles have been encountered to date.

This is the third of three parts of a paper by Mr. Lindberg, of the Oliver Iron Mining Co.

Mobile cranes on rubber tires are sometimes used in rail haul pits, but due to the usual lack of roads in such pits this use is limited.

Track-laying tractors are used for all kinds of dozer work, hauling, pushing; and with special hoist attachments, for hoisting, clamming, dragline, skull cracking; and with another attachment, can also be used for loading materials. They range in size from 4 to 25 tons, with 6% in the size range from 4 to 10 tons and used for light cleanup work, 11% in the 15 ton class, and the balance are tractors of 20 tons or over. The trend is toward still larger and heavier machines. Of the approximately 190 tractors on the iron

Rail Haulage Pits

ranges in Minnesota, 83% are the product of one manufacturer and feature heavy, rugged construction and powerful, slow-speed Diesel engines. These tractors have been in short supply. The most common and popular size is approximately 22 tons with the dozer, and, for blade lengths, from 11½ ft for straight blades to 13½ ft for angle dozers. Angle dozers are favored in rail haulage pits for grading track beds, dozing dumps, or whenever material is to be moved to either side. Angle dozers, of course, can also be set for straight dozing. The split on straight and angle-dozing tractors is 50-50 at the present time.

Blade control is hydraulic on approximately 85% of the large tractors; the balance are cable operated. The trend is back to more hydraulic control. Some recent hydraulic dozers have direct-acting, long-stroke cylinders which anchor to a special tractor subframe and attach directly to the blade. The advantage of hydraulic control is the ability to power down on the blade for greater digging effort. On cable-operated tractors digging effort is secured with the angle of the cutting lip on the dozer blade, but this does not control too effectively. An advantage of cable control and the new direct-acting hydraulic control is the close coupling between the tractor and dozer which cuts down the heavy unbalanced forward weight of the dozer which is hard on tracks and rollers when traveling.

Although tractors have been light in construction and performance and recent heavier designs are still in the development stage, one of the new heavier units incorporates a torque converter drive which has been quite successful to date. This trend in drives is in line with new developments in truck drives.

When stripping dump trestles have been filled to capacity, spreader plows push additional dumped material over the edge of the fill in order to widen the top to permit shifting the track

Twenty-ton mobile crane, with boom in traveling position, used at the Spruce mine.

Moving day for a meter house at the Oliver Iron Mining Co. Job is simplified by 20-ton mobile crane which loads house on trailer.

This 25-ton tractor with torque-converter drive is equally adaptable to grading track beds or removing heavy Minnesota snowfalls.

sidewise. When the spread fill is wide enough, a second track is laid and these tracks are kept shifted out to the edges of the fill, the tracks diverging until the dump area has been covered.

Spreader plows are made by two manufacturers and consist of heavy flatcar with a vee plow at the head end, two short free wings hinged to the car at the end of the vee plow, and two heavy wings hinged to the front end of the car about 5 ft from the end. These wings are about 25 ft long and extend at an angle of 30 to 45 degrees when in the plowing position. The plow is pushed by a locomotive and gets its operating air from the locomotive dump line.

Track Shifters—When the stripping dump had been filled and spread to capacity track shifters were formerly used to shift the track to the edge of the fill and then the dumping and widening program was resumed. These track shifters were in reality specially designed cranes with a tubular steel jib boom, a spacer boom of reinforced timber, and a wood and steel "A" frame, all mounted on a conventional railroad flatcar.

Standard steam railroad cranes were also used for track shifting and with either machine the method was to start at the outer end of the fill, hook on to the track, raise it clear, and boom it in the desired direction to the edge of the fill. The shifter then retreated, shifting as it did so, until the track was thrown.

Latest practice in track shifting makes use of tractors, with angle dozers fitted with special lifting hooks, that reach over and raise the outside rail on the hooks while the tractor shoves the whole track over to the edge of the fill. When the track is shifted, the same tractor dozes ballast over the inner end of the ties for use by the track gang when lining up and leveling.

Car Spraying—In winter operation some means must be provided for preventing stripping and ore from sticking to the cars. The idea popular in the past was the use of hot salt brine sprayed in a thin film on the inside of the clean stripping car. This was applied by a variety of methods, including hand spraying.

The use of salt has been largely superseded by substitution of calcium chloride in solution. This has lower freezing values and does not require any heat. The mixture is pumped from the tank house to a set of spray heads swung over the track that throw a fine spray directed against all parts of the inside of the car. The brakeman handles the control valve and the whole train is run through the spray at a speed of about 2 to 3 mph without stopping the spray. The whole operation of spraying a train of six cars is completed in about three minutes.

The solution in the tank house is adjusted for strength according to prevailing temperature and can be kept in continuous circulation by the same pump that supplies the spray heads. Spraying has also been done with Diesel fuel oil which can be handled with similar equipment.

During the past season another oil called "Spray Oil" has been used on the ore shipments to the docks. This is a light mineral oil about the equivalent of a hydraulic oil which is effective in preventing sticking in freezing weather, but becomes too thick for spraying at temperatures of 10 to 15 degrees above zero and has to be thinned with lighter fuel oil. It also has a higher delivered cost than fuel oil.

Common salt is the cheapest in cents per gallon but is the most expensive from the standpoint of the greater quantity required, the necessary heat, and the attendant labor. It is quite effective and is still used in some rail haulage pits. Calcium chloride is more expensive than salt but requires no heat and less labor. Both of the above are objectionable because of their corrosive nature. Fuel oil is the most expensive in cost per gallon but does not require any heat or attendant labor for preparation, is effective in smaller quantities per car, and is not harmful. Because of the importance of fuel oil in our present national economy, there is a question of the advisability of using oil for this purpose. The question of relative cost of each method is controversial and appears to be governed by the efficiency and effectiveness of the spraying equipment and the layout.

Tie Tampers—To expedite final finishing of permanent and semipermanent track, tie tampers are used for the tamping operation. These consist of rail cars mounting a two-stage compressor of about 105-cfm capacity at 100 lb pressure, and driven by a 32-hp, 1225-rpm gasoline engine. The unit is complete with pressure governor control and air receiver. Enough air is compressed to supply eight pneumatic tampers or spike drivers for use of the track crew. The unit is self-propelled with an air motor, chain coupled to one axle through reduction gears.

Other equipment includes a motorized power wrench for speeding up the tightening and taking apart of track bolts.

A new method of making holes in track rails for splice bolts uses a 45-70 caliber cartridge to shoot a punch through the rail web. This method is in use at our open pits but is not acceptable to the railroads for their track systems because of the stresses set up in the rail.

Speeders—In pits where track systems are usually the only connecting travel links between all parts of the pit, it is necessary to have some form of transport to get supervisory and maintenance forces to the places where they are needed and to move certain supplies that are required daily, and to do so as rapidly as possible. This is self-evident when it is realized that the Hull-Rust pit alone has 55 miles of track system.

Some of this transport can be handled with the regular haulage services, but for special and emergency use some form of carrier must be available to these agencies.

For movement of supervisory and maintenance personnel as well as equipment we have gasoline-powered motor speeders. The power units are automobile engines of about 65 hp with four-speed transmission and clutch similar to auto equipment driving the regular flanged car wheels on one axle. Simple mechanical brakes act on all four wheels. The car is covered with a housing and is large enough to carry the operator and nine men. There are also a few small speeders used by track crew supervisors to cover their territory. They are only large enough for the operator and a single passenger and are powered with a single cylinder, 8 hp water-cooled gasoline engine. Top speed is 30 to 35 mph on the level.

Diversification vs Unification in

MINERAL ENGINEERING CURRICULA

By William B. Plank

IN my studies during the past twenty years of the enrollments in the mining and metallurgical schools of the United States and Canada, I have been struck with the great diversity in the curricula offered by these schools.

In 1933 when I made my first complete study of the schools (*Mining and Metallurgy*, May 1934, p. 211), I discovered 19 different curricula or programs of study, leading to almost as many different bachelor degrees, in the 49 schools in the United States and the 4 schools in Canada. These were: (1) Mining Engineering, (2) Coal Mining, (3) Metal Mining, (4) General Mining, (5) Mining Administration, (6) Mining and Metallurgy, (7) Metallurgical Engineering, (8) Metallurgy, (9) Petroleum Engineering, (10) Petroleum Refining, (11) Petroleum Geology, (12) Petroleum and Natural Gas, (13) Ceramic Engineering, (14) Ceramics, (15) Geological Engineering, (16) Mining Geology, (17) Economic Geology, (18) Geophysics, (19) Fuel Technology.

Paper read before Middle Atlantic Section ASEE, Columbia University, Dec. 3, 1949.
The author is head, Department of Mining and Metallurgical Engineering, Lafayette College.

The number of schools has since grown to 64 in the United States and 8 in Canada, and in the report on the 1948-49 (*Mining Engineering*, August 1949) enrollment in these schools we find these additional titles to the programs of study: (1) Oil and Gas Engineering, (2) Physical Metallurgy, (3) Process Metallurgy, (4) Welding, (5) Mineral Dressing, (6) Mineral Sciences, (7) Mineral Engineering, (8) Mineral Economics, (9) Glass Technology.

Mineral Engineering education is not alone in this multiplicity of names. In the ASEE report on total engineering enrollment in the United States and Canadian schools for the past academic year, the tabulator lists 15 different curricula, including 4 in the mineral group. In addition, however, the total enrollment figure of 250,000 students in the two countries includes 73,000 whose courses of study are not specified. It can be assumed that the tabulator placed in this group all those whose courses he had difficulty in identifying with the 15 principal curricula he listed.

Merely because there is also a great variety of curricula in the *other* engineering branches is no excuse nor reason for us in the mineral branch to throw aside proper and reasonable restraint on unlimited division of *our* curricula. We should in fact be leaders to our fellow engineers in the matter of clear definition of the aims and objectives of our educational programs, and we should use terms in defining these programs that the mineral illiterates will understand.

The similarity in content and objective of many of the mineral courses listed above is so manifestly apparent that there seems to be little justification for this diversity in names and titles. Undoubtedly, these names developed because of different traditions at the various schools and because of the desire on the part of the educators to keep up with the unprecedented developments in the mineral industry during the past 50 years.

But this expansion in the number of programs has created lamentable cleavages between the various groups and a certain loss of cohesion and unity has resulted thereby. It also has resulted many times in confusion about our field of engineering in the minds of the general public and even engineers themselves, as will be described further on.

D. C. Jackson in his "Present Status and Trends of Engineering Education" (*ECPD Publication*, 1939) says about this matter, that "The dynamic changes which have been affecting all important engineering branches have tended to blunt the keenness of definitions of the purposes of engineering education as far as such existed. That is, the definitions of engineering education that are to be inferred from printed and spoken statements seem to grow less exact."

I wonder if we haven't been in error in trying to educate our young men in the colleges for separate branches of the mineral industry rather than for the **mineral engineering profession.** Have we not lost sight of the fundamental unity of the basic sciences underlying all branches of mineral engineering? Are we not, therefore, teaching special practices in these branches believing that our graduates may at once find responsible positions in specialized fields of the industry?

When we do this, I wonder if we are fully aware of the rapid changes taking place in practice—changes that more often than not make such practice obsolete by the time the young men reach the field. And, as perhaps the most harmful result of this specialized education, are we not preventing our graduates from taking advantage of opportunities for themselves in related fields and thereby robbing them of their fullest possible professional development.

Mineral Engineering educators who may be considering setting up specialized undergraduate

courses in mining should remember that a study of the SPEE showed that only about 50 percent of mining graduates find their life careers in the fields for which they prepared in college (Bull. No. 3—SPEE Investigation of Engineering Education, October 1926). What this percentage for the other branches of the mineral field is, is not known, but the study showed that 75 percent of the civil engineers remain in their chosen field.

T. T. Read, in his book "The Development of Mineral Industry Education in the United States", published in 1941 by the AIME, says that "the mining curriculum apparently has a larger transfer value than most other engineering curricula."

I submit then that we should not, by overspecialization of mining curricula, destroy the opportunities our graduates may have of profiting in other fields.

When one of our graduates wishes to take postgraduate work at another institution he may find himself barred from doing so if he has followed a too highly specialized undergraduate program. This is the situation which I believe is happening in the field of metallurgy today. For example, it seems to me to be difficult at present for an interchange of graduate students between the eastern schools and the western schools. I say this because of the different objectives I have found in the metallurgy schools of the two regions. Western schools are inclined to specialize in extractive, or process, metallurgy of the nonferrous metals, whereas the eastern schools because of their proximity to the large steel industries, are inclined to lean toward physical metallurgy and heat treatment, especially of ferrous metals and alloys.

An interesting case of how the nomenclature of various curricula in our field of engineering causes confusion was forcibly illustrated during the war when the Selective Service could not understand the different divisions of the field that we as educators have set up. I recall the confusion of an army officer, who met with the Education Division of the AIME on the Columbia Campus during the war, in trying to distinguish between a **mining engineer,** a **metallurgical engineer** and a **mineral engineer.** We all know that our students were not given equal consideration with other engineers in the matter of deferment from military service to finish their college work. Also, no provision was made for any mineral curricula in the Army Specialized Training Programs similar to those for civil, mechanical, and electrical engineers.

Another example of a confusion in names as applied to the divisions of mineral engineering came up about a year ago in the AIME when the Directors were revising the Institute's Bylaws. The section dealing with the qualifications of members contained this very ambiguous language: "he must have had at least six years employment in the practice of **engineering, mining, geology, metallurgy,** or **chemistry** ——."

When the Directors looked at these qualifications in the light of our present-day understanding of what engineering is they endeavored to remove in some way the ambiguity and indefiniteness in the statement and the inference that **engineering** is one thing and **mining, metallurgy, geology,** and **chemistry** are something else.

This section of the bylaws was revised to read: "A person to be eligible for election on transfer into the class of member must . . . have had at least six years employment in the practice of **Mineral Engineering,** or in the application of the sciences to any of the branches of the mineral industry."

Perhaps the most serious effects of these academic divisions within the mineral family are those taking place on our college campuses today. At two mineral engineering schools I know there are separate societies with national society affiliation for students in different mineral curricula.

This situation is lamentable because it destroys the unity and cohesion that has always existed in the past among our students. Further than that, it seriously impedes the beginning of the professional development of the students which membership in a common student engineering society so ably promotes. It is causing a serious defection on the part of our students of their allegiance to the AIME, which is recognized as the leading professional association of mineral engineers.

I fully realize, as you no doubt do, that I have here discussed two different kinds of diversification, namely: (1) diversification and subdivision of curricula within the broad mineral field, and (2) diversification of the names with which we label these sub-divisions. In my opinion both practices have their disadvantages, some of which I have explained in this paper.

Some of the evils of overspecialization in mineral curricula I feel will be rectified under the wise influence of the ECPD. And we, as educators and advisors to the students under us, should do all we can to bring about unity in student relations on the basis of professional fellowship and community of interests in all branches of Mineral Engineering.

One hopeful development in the simplication of names is the growing use of the term "Mineral Engineering" to signify the class or genus of all the specialized activities carried out by engineers in the mineral industries. Already educational institutions are beginning to use this term in this way in place of the old term "School of Mines". In its last catalog, the Colorado School of Mines used the connotation for itself of **A School of Mineral Engineering.**

Then finally, I suggest that most of the ambiguity and indefiniteness concerning our mineral curricula would be removed if all the varieties of such that I have mentioned in this paper would be placed in the following engineering groups: (1) Mining Engineering, (2) Metallurgical Engineering, (3) Petroleum and Natural Gas Engineering, (4) Ceramic Engineering, (5) Geological Engineering, and (6) Fuel Engineering.

I can foresee that as the mineral sciences continue to grow and expand there will be demand for new curricula—experimental in their early stages, at least—to teach these new sciences and their applications. However, for the sake of clearness I feel that these should be presented as options and not as major curricula.

Then it would seem logical to use the term **Mineral Engineering** in its generic sense to denote the groups as a whole. I have used the term in this sense in printed reports on enrollments in the schools since 1944. My prediction and hope is that the term will be thus universally adopted.

Exploration to Replenish World Mineral Reserves

by Walter Hull Aldridge

Much has been said about the foreseeable exhaustion of known mineral reserves—particularly in the United States. It is claimed that we are about to become a have-not nation insofar as certain essential minerals are concerned; and in this respect our position is often contrasted with that of the world as a whole, largely because we have drawn so heavily on our domestic resources during the last fifty years. As a matter of fact, the difference is not great if one takes a reasonably long-term view of the situation. No one denies that nature endowed the United States richer in mineral resources than any other equal area. It is granted that we have skimmed the cream of many kinds of minerals; but in a free international trade in minerals it would not be many decades before the richer deposits—at least the deposits easy of discovery—throughout the world would be depleted to much the same degree as those in the United States today. For it is a fact that the problem of satisfying the needs of our industrial society for minerals is fundamentally worldwide in scope.

This talk was given by Mr. Aldridge on the occasion of his receiving the John Fritz Medal for 1950 at the University Club, N. Y., Nov. 16, 1949.

The singular and significant attribute to all minerals is that there is but a single crop. Forests can be grown, but mineral deposits can be replaced only by finding new ones. Thus, one of the vital tasks of the mining engineer, the petroleum engineer, and the geologist is to find these new mineral resources. Another task is to devise means and methods for the exploitation of, at costs that are economic, low-grade deposits and those deeply buried in the crust of the earth. The first need, however, is to find the deposits in the ground.

If the assumptions of those who today fear an early exhaustion of all deposits are valid, the outlook for continued material and social progress is gloomy indeed. I take a more optimistic view, for it is my belief that vast new sources of needed minerals will have come to light before our known resources are gone. The real question is whether the outside layer of the crust of the earth that is accessible to engineers contains adequate mineral deposits; and whether engineers can find those deposits.

The answer must be based on geologic science, geologic fact, and mining history. Every continent, and almost every country, has yielded minerals of industrial utility in substantial quantity. Mineral deposits are widespread. Geologists know the conditions that were favorable to the occurrence of ores and the types of rocks in which various kinds of ores may ordinarily be found. They know that such rocks occur over broad regions of the earth's surface. They also know that in large areas rocks that are potential carriers of ores are covered and concealed by material of entirely different character. Such cover may consist of lava flows, layers of volcanic ash, or beds of material washed down from the surrounding mountains. Sometimes those covers are thick, sometimes relatively thin. But men well-qualified to pass judgment are sure that concealed deposits, large in number and in size, exist close enough to the surface to make exploitation practical. Only by actual penetration of the surface can this contention be proved, but to conclude otherwise would be to ignore the laws of probability and to deny the over-all consistency of nature.

The task of probing the crust of the earth to determine the location of these deposits, and to obtain some notion of their size and quality, poses serious problems. However, they should not be beyond human solution. Much more research must be done in analyzing and explaining the laws of ore deposition. New tools, in the shape of techniques and methods and of mechanical and electrical devices and instruments, must be developed. But no reasonable person can doubt that these objectives will be achieved. As the need becomes increasingly pressing, scientists and engineers will supply the imagination and the ingenuity that will provide the solution.

The time and effort thus far devoted to the development of methods for locating hidden ore bodies have been pitifully meager. Energetic investigation of the subject of applied geophysics is essential to the success of the mineral industry, and I have no hesitation in predicting that effective techniques will be found and successfully applied.

Twenty-five years ago the petroleum industry

was threatened with the early exhaustion of its known reserves. Resort was had to the relatively new techniques of geophysics. Tens of millions of dollars were spent; but the result was a replenishing of supply that exceeded all expectations, and today proved reserves are greater then ever before. Those of us engaged in the mining of metals and other industrial minerals should adopt the attitude so well expressed recently by the head of one of the largest oil companies: "As oil from a given field is used up new fields must be found. We know there is enough oil in the world for generations to come; but we would not consider ourselves as meeting our obligations unless we made every effort to continue to locate this oil." The great advances made by the petroleum geophysicist in developing suitable tools should spur the metal miner to similar successes.

It is probable that the unit cost of some essential metals and minerals in terms of labor, power, and capital equipment will increase. But here again the ingenuity of the engineer may bring surprising results. It is true that the noteworthy strides of the past fifty years have narrowed the area in which further improvements and economies are possible. Block caving underground, and improved methods of open-pit mining, coupled with differential flotation and other ingenious processes of ore treatment have not only turned huge masses of mineral-bearing rock into ore, but have made possible costs that seem almost fantastic. When the percentage of contained metal recovered from an ore has been increased to 95% (compared to 60% a few decades ago), obviously the margin for further increase is slight. When ore is being drilled, blasted, and loaded into cars for 20¢ per ton (as compared with $1 not so long ago), the scope for improvement is not large. Accordingly, the field for further progress must be in the perfection of techniques for exploiting mineral deposits that are of progressively lower grade and that lie at progressively deeper horizons below the surface of the earth.

I believe that exploitable deposits in the world are far larger than many students of the subject seem to think. Mining engineers, with the indispensable help of physicists and engineers in the other branches of the profession, will find these deposits and deliver the products to industrial users in such quantities as may be required. To doubt this is to underestimate the ability of the members of the great engineering profession.

New Drawing Practice at Braden

BECAUSE it was believed by staggering raises between the grizzly level and the undercut level on opposite sides of the drift there would be less chance of having pillar failure between these levels, we have been required to draw a large proportion of ore into one-ton cars and tram a short distance to the grizzly. Fig. 1 is a plan view of this setup through the grizzly level, showing ore passes through which the ore was transferred.

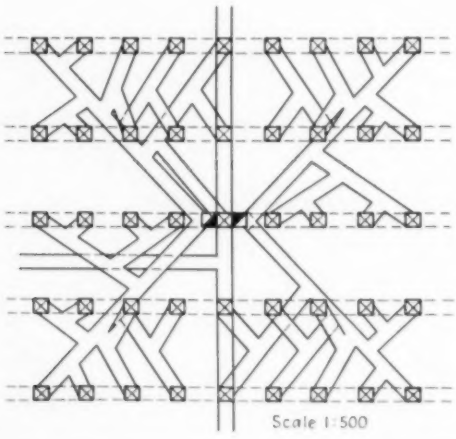


Fig. 2. Present set-up has eliminated tramming by putting more transfer raises up to grizzly level to break through beneath draw points.

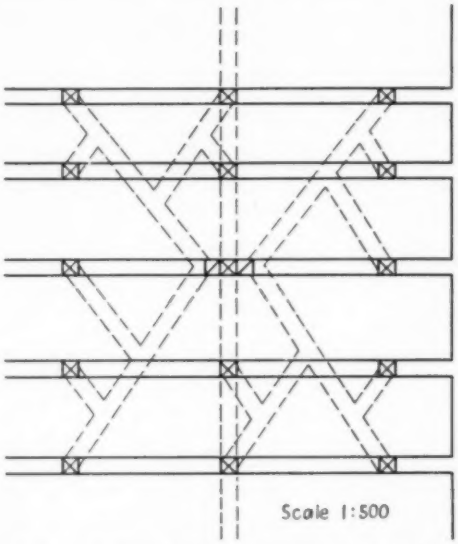


Fig. 1. Ore must be hand trammed to transfer raises at the grizzly level in this arrangement.

Tests, however, have shown that the difference in pillar strength is not as great as was anticipated so that raises are now being driven opposite each other and adapting the ore-passes below to connect under each pair of draw points—thus doing away with all hand tramming (Fig. 2). This naturally makes for cheaper operation although it requires much more development work.—**F. E. Turton.**

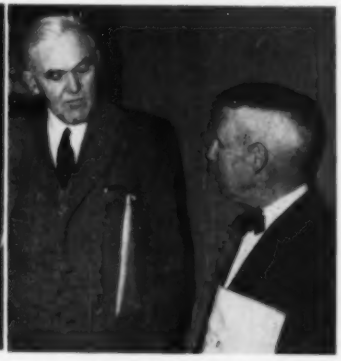
Annual Meeting Album

OUR 78th Annual Meeting was, technically and attendance-wise, one of the greatest to date. No one who came to New York's Hotel Statler had time to hear all 266 papers, or to attend all 70 sessions and 27 committee meetings, or to shake the hands of the more than 3000 engineers, educators, students and executives who attended—but no one was lost in the crowd. For crowds always dwindle down to groups of two and three and four, and those are the little sessions which make our meetings well worth attending. Herewith four pages of photographic highlights and sidelights on the February gathering.

(Photos by Ken Henderson, courtesy of International Nickel Corp., and by Gerry Adams.)

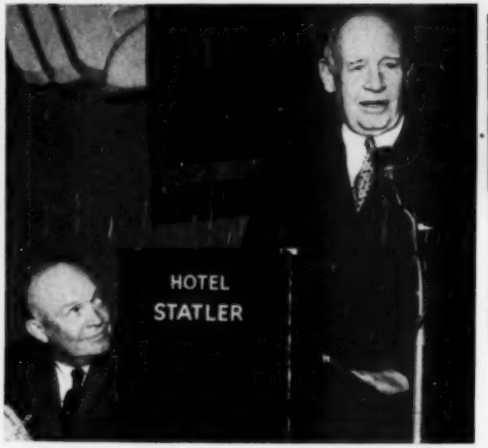
N. L. Shepard, left, Chairman of the St. Louis Section this year, watched things carefully, made mental notes for next year's gathering when St. Louis will play host. The USGS' H. M. Bannerman (below left) sought a quiet corner to check his speech before addressing The MIED on "Training Geologists."

The kickoff day brought many a smile as old friendships were renewed and new ones made. Our roving photographer caught (top LtoR) W. A. Staab, Roy H. Smith of Mid-Continent Section, F. W. Strandberg, Montana, Roland Gouldy from the North Texas Section, and E. H. Rose, at the Council of Section Delegates. Below are P. P. Ribotto, Upper Penin.; J. L. Weaver, Fla.; V. J. Mercier, Kan.; and H. P. Sherman, Columbia.



Young hopefuls like R. L. Folkman, A. H. Moon, R. L. Speer, and J. M. Jenkins, who came down from Cornell, mingled near the registration desk with H. M. Roberts, consultant from Duluth, and Life Member C. W. Knight of Toronto (right).

GENERAL DWIGHT D. EISENHOWER, the man who engineered the defeat of the Axis, and who now has a big hand in the future as president of Columbia University, spoke at the Welcoming Luncheon after meeting AIME President L. E. Young, and our incoming President, D. H. McLaughlin. The General cautioned against the gradually increasing power of the federal government, said: ". . . freedom as we know it is in no greater danger from without than it is from our own lack of understanding of this creeping paralysis from within," and recalled the image of the prospector on his burro who was "only vaguely aware of who was President and what was going on in Washington."



The General (photo at left) listened with obvious pleasure to the noted Cornelius F. "Con" Kelley, chairman of the board of the Anaconda Copper Co., and then rose (above) to captivate his audience by leading off with a tall tale from Kansas. Drs. Young and McLaughlin and Rep. C. A. Herter of Massachusetts are shown enjoying "Ike's" humor. Other luncheon speakers were Fred V. Siebert, president of CIM, and W. A. Staab of the Calcium Chloride Ass'n.

William H. Humphries and John A. Harrison, who won $100 student prize paper contest awards, are congratulated on their achievements by President McLaughlin. Awards were presented at the Welcoming Luncheon.

Another distinguished speaker was Massachusetts' Rep. C. A. Herter, shown addressing the Mining and Metallurgical Society of America.

Father J. B. Macelwane, Dean of St. Louis University, elicited careful attention from Associate Chairmen George M. Fowler and Sherwin F. Kelly as he read a paper before a joint session of the Geology and Geophysics Subdivisions.

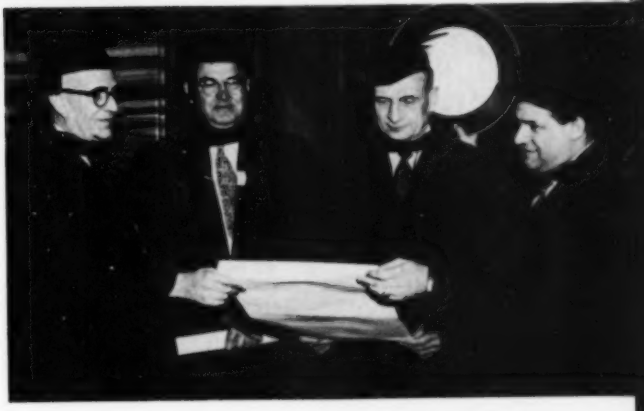
D. N. Griffin of the Deister Co. and W. J. Parton of Lehigh Navigation Coal acted as Associate Chairmen of the session on Fine Coal Cleaning on Tuesday.

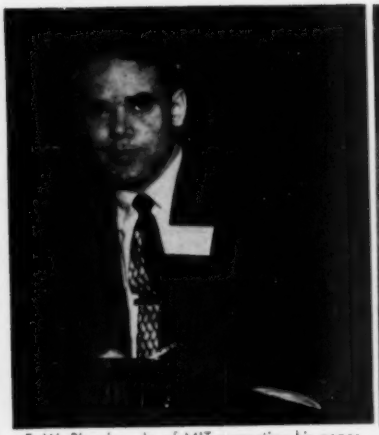
Taking over the reins of the MBD, Grover Holt of The Cleveland-Cliffs Iron Co., Ishpeming, Mich., (left) got a hearty sendoff from S. J. Swainson, the retiring chairman, who's director of ore dressing for American Cyanamid in Stamford, Conn.

Business with a smile at the MBD business meeting. F. W. McQuiston, Jr. of the Atomic Energy Commission, Harlowe Hardinge, and W. B. Stephenson of Allen Sherman Hoff did the smiling.

Over 300 members of the WAIMME added their charming presence to the meeting, unpacking satchels full of interesting social and cultural activities. This group at the registration desk includes (L. to R. from the top): Mesdames Dan Young, F. T. Kett, W. C. Bowden, Jr., E. H. Robie, F. E. Wormser, D. H. McLaughlin, J. P. Bonardi, H. Y. Eagle, W. N. Best, C. H. Eichrodt, J. F. Kemp, and Mrs. D. Moreno.

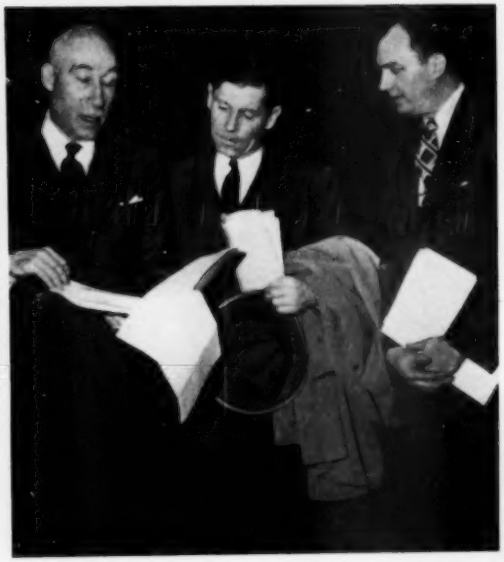
Paeans of pedagogic praise were heaped upon Ted Counselman, E. H. Rose and J. F. Myers at the MBD luncheon as Don Scott (right) conferred the "Sears Roebuck degree" of "Mill Gentleman of Distinction" on them. The prospector's pan added lustre to the proceedings, and allowed the gentlemen to bask in their own reflected glory for a while.





F. W. Bloecher, Jr. of MIT presenting his paper at the MBD session on Monday afternoon.

At the Council of Section Delegates were Eugene McAuliffe, Curtis Wilson, Dean, Missouri Sch. of Mines; H. I. Young, president, Amer. Zinc, Lead and Smelting.



Other Section Delegates were Dean M. I. Signer, Colorado Sch. of Mines; Roland D. Parks of the MIT, Linwood Thiessen of Pittsburgh.

Entertainment committeemen and their friends enjoyed the big show they arranged for the annual Dinner-Smoker in the Ballroom of the Statler. L. to R., H. J. Picardo, Jr., A. T. Ward, J. P. Bonardi, chairman; G. B. Higgins and Miss Gwen Beaugod.

At the Student Careers Conference, good advice over a glass of beer was given to students H. Brandt, George Mendel, Spencer Shannon and Bill Wood by President McLaughlin. Several other educators and working engineers contributed to an interesting session.

Exclusive closeup of earnest students of square dancing attending a "technical session" at the informal dance. Some hardy enthusiasts even had enough surplus energy to enter the waltz contest which followed.

Use of Concrete Underground

Use of concrete underground at No. 4 mine, Cornwall, Pa., has become increasingly important. Starting with small amounts at finger openings, occasional raises, piers, its use has expanded to entire slushing drifts and haulages. Besides eliminating costly timber repairs, concreting costs of slushing drifts have proved cheaper than timbering. Higher haulage-concreting costs are offset by no maintenance charges during the life of the haulage.

by Joseph Bernhardt

THE Cornwall Ore Mines, Division of the Bethlehem Steel Co., at Cornwall, Lebanon County, consists of two separate magnetite ore bodies, approximately one mile apart. The one ore body was an outcrop in which open-pit mining operations were begun in 1742 and continued without interruption. This mine is also served by a shaft for mining that portion of the ore body not recoverable by open-pit methods.

The other property had no outcropping and is entirely an underground operation. This mine, captioned No. 4 mine, is serviced by two shafts 220 ft apart. The No. 4 shaft is for hoisting rock and ore, and the No. 5 shaft is for handling men and materials. The shafts are on a catenary curve, closely following the dip of the ore body. Slope of the shafts at the collar is 36°, flattening to 26° at the bottom.

The footwall at Cornwall is a traprock which varies structurally from solid to highly fractured. Grizzly stations at the top of each chute raise were reinforced by setting timber sills on concrete. Occasionally it was necessary to concrete the entire chute raise. Concreting was done by using ½-yd electric mixers set up in the breaking subs. Sand, stone, and cement were hoisted up through the chute raises to the breaking subs.

Late in 1937 it was decided to eliminate chute raises and breaking subs in future development. Haulage levels are 50 ft apart vertically. When ore is below a 40° dip, slushing drifts are driven normal to the haulages and on 40-ft centers. In a few areas, ore dips greater than 40° and slushing drifts are then placed parallel to the haulage drifts. Finger raises are opposite each other and on 20-ft centers making 20x20-ft draw areas (fig. 1). For loading 90-cu ft cars, 50 hp, two-drum scraping hoists and 54-in. scrapers are used. The lengths of slushing drifts vary from 90 to 200 ft, depending on dip of the ore.

The slushing drifts were timbered with 10x10-in. posts and 10x12-in. caps. Sets were spaced to make three spaces 4 ft-8 in. and one space 6 ft center to center of timber. Two additional framed sets were then placed inside the 6-ft space to make the finger station. The sides of the fingers were concreted to 12 in. above the drift timber and a heavy round timber placed on top of the concrete which acts as a bumper cap.

JOSEPH BERNHARDT is Superintendent, No. 4 Mine, Bethlehem Steel Co., Cornwall, Pa.

AIME Columbus Meeting, September 1949.

TP 2830 A. Discussion (2 copies) may be sent to Transactions *AIME before April 30, 1950. Manuscript received Aug. 22, 1949.*

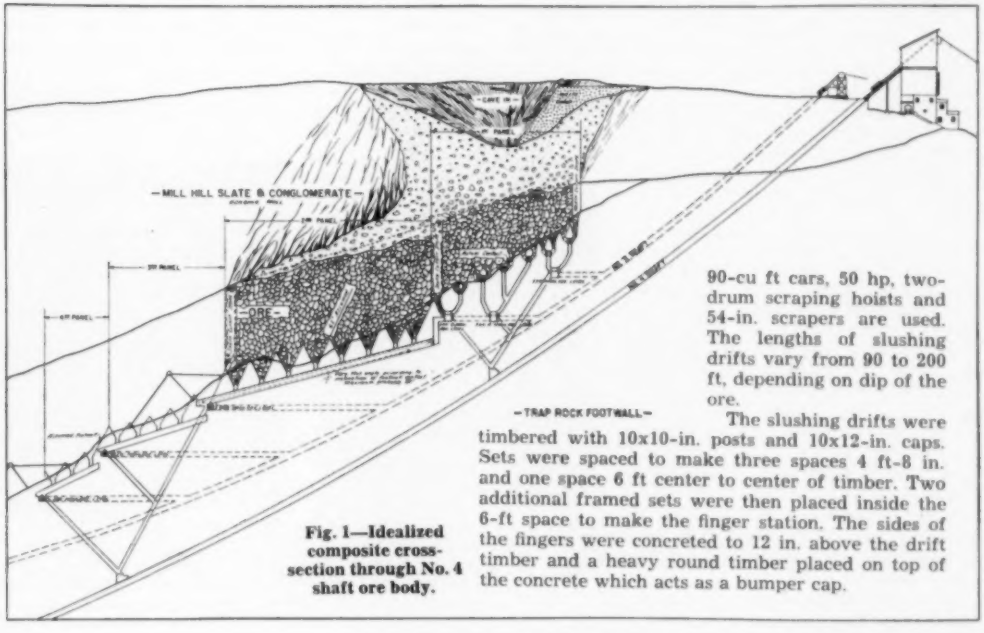


Fig. 1—Idealized composite cross-section through No. 4 shaft ore body.

This page of Transactions AIME is a continuation from p. 296. The missing pages appeared in Mining Engineering.

Until 1943, all concrete was mixed at the mine. Since then, ready-mixed concrete delivered from Lebanon, which is about 6 miles away, has been used exclusively. Other uses of concrete were for the shaft back, level landings and ore-pocket raises. A pneumatic ball type concrete placer was used for placing concrete in the shaft back and level landings.

Concreting Slushing Drifts: In all cases, timbered slushing drifts had to be repaired before being drawn empty. The timbering in many of the drifts had to be completely replaced and sometimes second repairs were required. (Fig. 2 shows a slushing drift that required extensive repair.)

In 1945 it was decided to concrete slushing drifts. It was expected that the initial cost of concreting against cost of timbered drifts would be higher, but if timber repairs were eliminated, considerable savings would result. The concrete would also eliminate the hazardous work of timber repair.

The following equipment was ordered and received in November of 1945:

1—¾ cu yd pneumatic concrete placer	$3,250.
1—lifting frame	400.
7—¾ cu yd concrete hoppers	770.
7—5-ft sections collapsible steel concreting forms including 4 travelers	4,025.
1—20-ft rubber 6-in. concrete hose	400.
2—90° shooting boxes	300.
Total	$9,145.

The linings of the first drifts were poured solid without finger openings. Mining of the finger openings through concrete was costly and it was decided to form out the finger openings.

The slushing drifts are mined 7 ft-6 in. wide and 8 ft-9 in. high. If the rock is good, fingers are mined in the ribs of the drift. Three 40-lb rails on wood ties are installed in the floor of the drift. The two outside rails are placed on 30-in. gauge to suit the carrying travelers of the steel forms. The three rails, later, make the scraper bottom of the drift.

Four, 5-ft sections of form placed on two travelers are used for each concrete pour. With fingers on 20-ft centers the joint is placed between fingers. The finger openings are formed using prefabricated wood on the side arc of the steel forms. Sides of the finger openings are concreted so that restriction is at the inside of the drift (42 in. wide by 54 in. high). A rail is placed across the top of the opening to give added strength; no other reinforcing is used.

The concreting cycle consists of pouring 20 ft one day, stripping and moving the forms up the next day, and pouring again the third day. Concreting is usually carried on in two adjacent drifts or two drifts on separate levels. This allows a 20-ft pour every day. The minimum thickness of concrete is 12 in. on the top and 9 in. on the sides.

Photographs illustrating some of the slushing drift concreting work are as follows: fig. 3 is a view looking up a 30 pct grade with forms in place, fig. 4 is looking down a 30 pct grade showing the pipe arrangement and forming at the back end.

When the first drifts were concreted, the placer was set up underground as close as possible to the job to be poured (fig. 5). Concrete was taken down the shaft in ¾ cu yd hoppers. This procedure kept our material skips too busy and it was decided to set the placer on the surface. A second placer was purchased and set up on the level in which slushing drifts were being poured. A 6-in. pipeline was placed in the manway of our No. 5 shaft. This line at present runs to our 820-ft level and has take-off pipes at the 650, 700, 760 and 820 ft levels. (Level designation is vertical distance below shaft collar.) The placer underground handles slushing drifts on two haulage levels, blowing uphill to one and downhill through manways to the slushing drifts on level below.

Fig. 2—Slushing drift that required extensive repairs.

Later it was decided to try using only one placer on the surface, blowing direct to the forms. In a slushing drift which was concreted recently, the concrete traveled 1350 ft down the shaft, 1316 ft on the haulage level and through a vertical distance of 30 ft to get into the upper end of the form, or a total of nearly 2700 ft. At this distance the concrete was somewhat dry and showed some segregation using a 1-2½-3½ mix. A mix of 1-2-3 gave better results. Sand and stone are crushed limestone. Per bag of cement, 4½ gal of water is used, with a wetting agent added.

Three sets of costs on completed concrete-lined slushing drifts are available to date and are given in table I.

Fig. 3 and 4—Slushing drift concreting work.

Fig. 3—View looking up 30 pct grade with forms in place.

Fig. 4—View looking down 30 pct grade showing piping arrangement and forming at back end.

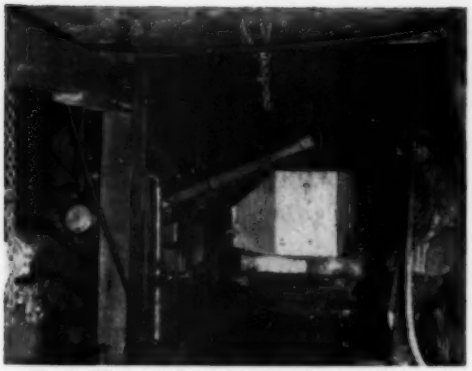


Fig. 5—View of placer setup underground close to the job to be poured.

Table I. Costs of Concrete-lined Slushing Drifts

	1 550-ft Level 2 drifts	2 600-ft Level 4 drifts	3 600 and 650-ft Level 14 drifts
Lineal feet	162.0	335.5	1,388.5
Total manshifts	271	504.5	1,607.5
Cubic yards concrete	199	360.5	1,910.5
Manshift per foot	1.67	1.50	1.16
Cubic yards per foot	1.23	1.07	1.38
Labor cost per foot			13.73
Material cost per foot			14.15
Total cost per foot concrete			27.88
Total cost per cu yd concrete placed			20.20

(1) First two drifts were poured solid and were somewhat experimental causing high labor cost.

(2) Next four drifts were using two placers and included forming and concreting the finger openings. The first six drifts were 7 ft wide by 7 ft 6 in. high over top of rail.

(3) The size of the drifts was reduced to 6 ft wide by 7 ft high over top of rail. Ten of these fourteen drifts were poured using one placer on surface. Labor rates averaged $11.84 per manshift. Concrete at present costs $10.75 per yard at the shaft collar.

The average cost per foot of timbered drift including the cost of pouring finger opening by hand was $33.45 per foot as compared with $27.88 per lineal foot of concrete lined drift. Concreting the slushing drift has actually made a saving of $5.57 per foot.

Concreting Slushing Drifts in Heavy Ground: A very good example of using concrete in heavy ground occurred in 1948. Three 100-ft slushing drifts were driven in ore and limestone on 30-ft centers. Ten by 10-in. posts and 10x12-in. caps were placed on 2½-ft centers. The timber was placed while driving. Upon completion of the drifts, but before mining was started, the timber failed (fig. 6). The timber was taken out and rail sets were installed for a distance of 25 ft. Concrete forms were then placed (fig. 7) and a 24-ft section was concreted. This process was repeated until the drifts were finished (fig. 8). The inside size of these drifts was 4½ ft wide by 6 ft high. Special forms were used in this location because the forms had to be taken up 4-ft diam raises. The concrete held up well permitting 170,930 net tons of ore to be drawn from these three drifts.

Fig. 6—Timber failure after drifts were completed and prior to mining.

Fig. 7—Rail sets installed in place of timber and concrete forms placed.

Lining of Ore Pocket Raises: The concreting of ore pocket raises has been a major project at Cornwall. Each ore pocket serves three haulage levels. The ore pocket raises consist of a main raise 200 ft long and two branches. Raises were driven on a 55° slope and enlarged to 7 ft. Steel tubes 5-ft in diam made up of 1-in. steel plate on the bottom and ½-in. plate on the top were concreted into place. These wore out when approximately half the required tonnage had been drawn. The steel liners which were in 4 ft long sections loosened after the bottoms were worn through. This caused many dangerous hangups.

Fig. 8—Completed drift with rail sets and concrete forms.

Fig. 9—View of concrete form in longest raise.

Fig. 10—Concreted haulage drift and intersection.

The latest loading station located on our 880-ft level has three separate raises emptying into a scraping trench 90 ft long. The scraping trench is 10 ft wide and 14 ft high. These raises will service the 700, 760 and 820-ft levels. It is expected that elimination of the branch raises and resultant lower tonnage through each raise will enable the concrete linings to handle the required tonnages. Fig. 9 is a view of the concrete form in the longest raise (200 ft) and shows the very weak rock structure. A rail ladder was used for alignment of 24 ft of collapsible 5-ft diam forms. An aggregate of trap rock and silicon sand was used instead of the usual limestone aggregate. The concrete was a minimum of 18-in. thick on the bottom and 9-in. on the top and sides (mix 1-2-3).

Concreting Haulages: During late 1948 and 1949 some work was done on concreting haulage levels. Although 650 ft has been concreted to date, costs were kept on only 150 ft. This 150 ft of concrete was placed in 240 ft of haulage driven with an arched roof section, while the balance was driven with a rectangular section for conventional timbering (table II).

Table II. Cost of Concreting Haulage Drifts

Lineal feet	150.
Total manshifts	85.
Total cu yd placed	232.
Manshifts per foot	0.57
Cu yd concrete per foot	1.55
Labor cost per lineal foot	6.75
Material cost per lineal foot	16.66
Total cost per lineal foot	23.41
Total cost per cu yd placed	15.10

Concreting of the haulage consists of lining 25 ft of drift and leaving 15 ft unlined for the slushing-drift loading station. This is repeated throughout the length of the drift. These unlined 15-ft sections are temporarily supported with one timber set blocked to the concrete. This timber is removed when loading stations are required.

For concreting haulage drifts there is very little hand forming required. The ends of the sections are closed off and in the center of each 25-ft section poured, a man safety station 3 ft wide by 2 ft deep by 6 ft high is formed. The size of the concreted drift is 8 ft wide by 7 ft-6 in. high over the top of the rail. Fig. 10 shows some completed haulage drift in addition to a concreted intersection.

The following several advantages in addition to those mentioned previously have become apparent as this work in concreting haulage and slushing drift progresses.

1. Less experienced timbermen are required for both original placement, and occasional repairs to finger openings.

2. Development of haulage and slushing drifts can be far in advance of that work which is actually required with no deterioration of the development workings. This can be especially advantageous during shutdowns.

3. The driving of both slushing and haulage drifts with an arched section has permitted the drifts to stand unsupported until they could be lined with concrete.

4. The fire hazard from timber is practically eliminated.

It is interesting to note that the same forms can be used for either small size slushing drifts or larger haulage drifts by changing the bottom plate of the form. When expanded for smaller width as it is used in the slushing drift, the sides are not vertical, giving a slight horse-shoe effect.

In conclusion, a few figures regarding yardage and costs are given. From January 1948 to the end of June 1949, 3752 ft of haulage and slushing drift was concreted using 5314 cu yd of concrete. No. 4 shaft extension and miscellaneous construction underground required an additional 2850 cu yd or a grand total of 8164 cu yd of concrete.

Frequently more than one job is poured during the same shift. A record daily pour of 92 cu yd was made including the blowing of 55 cu yd from the surface to two different levels and 37 cu yd taken into the mine in ¾-yd buckets to a placer setup underground and blown to the scraper trench construction.

To maintain our expected monthly tonnage, 210 ft of slushing drift and 80 ft of haulage must be concreted per month. This has easily been attained, since as much as 420 ft of haulage and slushing drift have been poured in a month.

At No. 4 mine an average level 2000 ft long would have 6000 ft of slushing drifts, requiring 10,200 cu yd of concrete costing $196,500 in place. Based on an average height of 90 ft of ore, the cost for concreting the haulage and slushing drifts would be 8.2 cents per ton of ore.

Radiotracer Studies on the Interaction of Dithiophosphate with Galena

by G. L. Simard, J. Chupak, and D. J. Salley

Radiotracers were demonstrated to be of considerable value in a study of the interaction of dithiophosphate with galena. The interaction had characteristics of both chemisorption and chemical reaction, being complex because of the nature of the galena surface. Sorption was rapid and markedly reversible, and flotation occurred at relatively small amounts of dithiophosphate sorbed.

DITHIOPHOSPHATES and xanthates are the principal collectors for sulphide minerals, and consequently any knowledge of mineral-collector systems of this type is of value. In the present investigation an attempt was made to obtain information on the interaction of a typical dithiophosphate with galena. In carrying out this study, radioisotopes, so much discussed in the past few years,[1, 2] were extensively employed. By using radioactive dithiophosphate synthesized from radioactive phosphorus, a rapid and sensitive analytical procedure was at hand. This permitted determination of such important quantities as the rate of uptake of dithiophosphate by the mineral, the amount existing at equilibrium on the mineral surface and in the solution, and the desorption of the agent from the surface. Such measurements were possible even on single crystals of galena. In addition the exchange of dithiophosphate between the solution and the sorbed phase was examined, a matter which could be accomplished only by the use of isotopes.

It is desired to point out at this time that throughout this paper the term "sorption" has been employed to designate the uptake of agent by mineral, without implication as to the nature of the process by which the uptake was accomplished.

G. L. SIMARD is in the Research Division, Stamford Research Labs., American Cyanamid Co., Stamford, Conn.; J. CHUPAK, formerly in the Research Division, Stamford Research Labs., is now at Camp Detrick, Frederick, Md.; and D. J. SALLEY is in the Research Division, Stamford Research Labs.

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Experimental

Materials: *Galena:* The galena was from the Tri-State district. For studies on ground mineral four preparations were used during the course of the investigation. These were prepared from selected large crystals by wet grinding in order to reduce surface oxidation. Alcohol was chosen as a convenient medium for this purpose. The ground mineral was then fractionated by sedimentation in alcohol, dried by evacuation, and stored in a nitrogen-filled desiccator. Data on the preparations are tabulated in table I. The size analysis indicated that the areas of the preparations were of comparable magnitude, even though the absolute values may be somewhat incorrect.

Dithiophosphates: Nonradioactive dithiophosphate (di-isopropyl or di-secondary butyl) was obtained by purification of a commercial product. An aqueous acid solution of the agent was extracted with petroleum ether, the ether layer dried, and the dithiophosphate precipitated as ammonium salt with anhydrous ammonia. Several repetitions of this process resulted in a nearly colorless, flaky product of good purity. (General formula $(RO_2)PSSNH_4$).

Radioactive dithiophosphate was synthesized by heating radioactive elementary red phosphorus* with sulphur at 270° to 300°C to produce radioactive P_2S_5. The latter was then treated with the desired purified alcohol (isopropyl or secondary butyl) at 60° to 80°C to form the dithiophosphoric acid derivative. Purification was effected in the same manner as for the nonradioactive material. The effectiveness of the purification method was established by the isotopic dilution method.[3] The same technique was used to show that only negligible decomposition of neutral or of carbonate solutions of the dithiophosphate took place over a period of a day; this was true whether or not galena was suspended in the solutions.

* For early experiments (1943 to 1944), the elementary red phosphorus was obtained from the cyclotron group at the Crocker Radiation Laboratory of the University of California, Berkeley, through the courtesy of Dr. Joseph W. Hamilton. For more recent work (1946 to 1949), the elementary radioactive phosphorus has been supplied by the Oak Ridge National Laboratories on allocation from the U. S. Atomic Energy Commission.

Procedures: *Radioactivity:* The activity of a solution was obtained by counting with a small glass-jacketed, silvered Geiger counter, using a conventional scaling circuit.[4] Crystals and other solids were counted under a bell-shaped, mica-window counter tube. Sufficient counts were made so that the probable error of counting was of the order of ±1-3 pct.

The specific activity in terms of counts per minute

Table I. Data on Galena Preparations

Galena No.	Size, Bulk by Volume Micron	Area* cm² per g	Pb, Pct
2	2.5-20	1,800	85.7
3	10-30	950	85.1
4	10-20	1,130	85.7
5	5-25	1,100	86.0

* By Fischer sub-sieve sizer, except for No. 2, which was from microscopic size analysis. Cu, Fe, and Ag below 0.01 pct in all cases.

per mol was measured by determining the activity of a solution of known concentration. For solids, the specific activity was obtained by drying and counting an aliquot of a solution of known concentration. With the counters used, the specific activity of the most active dithiophosphate preparations was of the order of 10^{+10} counts per minute per mol, thereby allowing detection as low as 10^{-12} mols.

Sorption: To determine the uptake of agent by ground mineral, a weighed sample (0.85 to 1.0 g) was shaken in air for a specified time with a given volume of solution of known initial concentration of the radioactive agent. The mixture was then centrifuged (which presumably stopped the sorption process), the clear supernatant solution was pipetted out, and then was drawn through a cotton wad into the counter tube. The observed count and the known specific activity of the radiodithiophosphate allowed calculation of the residual concentration of the dithiophosphate. The difference in amount present before and after the experiment gave the quantity of agent taken up by the mineral.

In certain instances, the ground mineral sample was leached with water before a sorption measurement. The leaching was carried out merely by shaking with 30 to 50 cc of distilled water for a 30 to 60-min period, centrifuging, and pipetting off the supernatant as completely as possible. The sorption procedure was conducted immediately without intermediate drying of the galena.

Although most of the runs were carried out in the air, some trials were made in nitrogen. In these cases, boiled solutions stored under nitrogen were used, and the air in the dead space of the shaking tubes was displaced by nitrogen.

For sorption studies on galena single crystals, well-formed crystals with faces about 1 to 2 cm square were selected. A crystal was freshly cleaved and both halves were immediately immersed in a

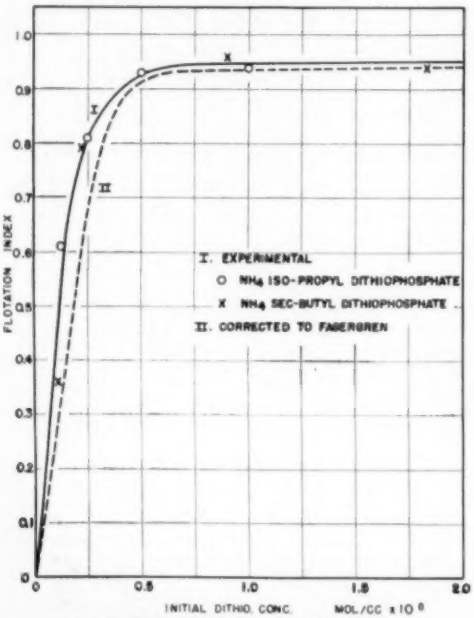

Fig. 1—Flotation curves of galena No. 2.
(20×10^{-6} mol per cc Na_2CO_3).

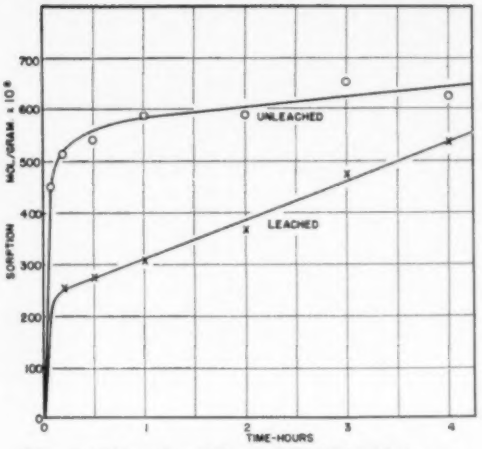

Fig. 2—Rate of uptake of dithiophosphate on leached and unleached galena No. 5.
Initial concentration: dithiophosphate—30×10^{-6} mol per cc. Temperature, 25°C. (20×10^{-6} mol per cc Na_2CO_3).

solution of the radiodithiophosphate for a specified time, after which they were removed and sponged dry immediately with Kleenex tissue. Crystals which gave irregular faces or cleavage were rejected. By use of a lead mask having a central hole, a nominal area of crystal was defined for count. Since the amount of material taken up by the crystal was relatively small, no sensible decrease in the concentration of the solution occurred.

Desorption: For desorption measurements with ground mineral the uptake of radiodithiophosphate was first carried out by the usual procedures, and then immediately a given volume of sodium carbonate solution (20×10^{-6} mol per cc) was added to the moist galena. After shaking this for a period, the amount of dithiophosphate desorbed was obtained from the count which appeared in the carbonate solution.

In measuring desorption from single crystals, the crystal bearing the radiodithiophosphate was merely submerged in water for a specified time, removed, dried with Kleenex, and counted.

Exchange: For an exchange measurement* between dithiophosphate sorbed on galena and dithiophosphate ions in solution, the usual procedure for measuring the consumption of dithiophosphate was first performed. After determining the concentration of the radiodithiophosphate solution left from the sorption run, a nonradioactive solution of identical concentration and volume was added to the galena. This mixture was shaken for a specified time and the count appearing in solution was measured. From the knowledge of the amounts of dithiophosphate present in each phase, a calculation was made of the count expected in solution if statistical distribution of the radioactive molecules between the sorbed and the solution phase had taken place. The ratio of the observed count to that expected gave the so-called "fraction exchanged."

In carrying out exchange between lead dithiophosphate and dithiophosphate ions, a freshly precipitated lead dithiophosphate was first formed by mixing lead nitrate with radiodithiophosphate. The

* For a general discussion of exchange, see p. 250 of ref. 1.

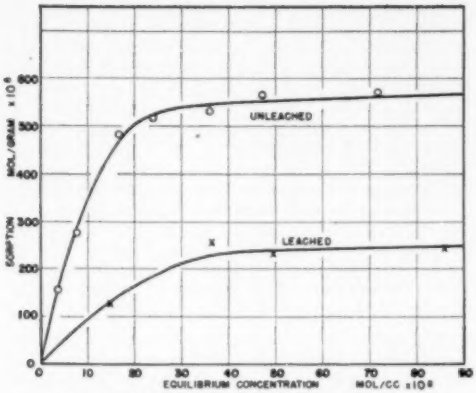

Fig. 3—Sorption isotherms of dithiophosphate on leached and unleached galena No. 5.

12 min sorption time. Temperature 25°C. (20×10^{-6} mol per cc Na_2CO_3).

washed precipitate was then suspended in a solution of known concentration of inactive dithiophosphate, stirred for a specified time, centrifuged, and the count which had appeared in the supernatant determined. Calculation of the fraction exchanged was again made by comparing the observed count with that expected on the assumption of complete statistical distribution of radioactive molecules between both phases.

Flotation: For flotation experiments a pneumatic flotation cell designed to operate on one-gram samples was constructed, using a fritted-glass disk as the air-dispersing element in a manner similar to that described by Herd and Ure.[8] The cell above the fritted disk was 3 cm in diameter and 5 cm in depth. In making an experiment, 25 cc of the solution to be tested was placed in the cell with the gram sample of the mineral. (On certain occasions 0.4 ppm of octyl alcohol was added as a frother.) A slow stream of air, insufficient to cause overflow, was then passed through the mixture for a conditioning period (usually 10 min). The air velocity was increased in order to produce flotation and the floated mineral was collected on a filter disc, dried, and weighed. Flotation was expressed by "flotation index":

$$\text{F.I.} = \frac{\text{weight of mineral floated—weight of blank floated}}{\text{weight of sample—weight of blank floated}}$$

The relationship between the flotation index values for the one-gram pneumatic cell and for a 100-g Fagergren machine was determined by making comparative tests. Fig. 1 shows typical flotation curves for the small pneumatic cell, as obtained experimentally and as corrected to conform to the Fagergren.

Results

The following sections outline the results obtained on the rate of uptake of dithiophosphate from solution, on the isotherms, on the desorption, and on the exchange, and include work on both ground mineral and single crystals.

All measurements were made in solutions buffered with 20×10^{-6} mol per cc of sodium carbonate, unless otherwise specified. The pH was constant at 7.8 ± 0.1 over a dithiophosphate concentration range of 0.5 to 75×10^{-6} mol per cc. It is of interest that tracer measurements made with radiocarbonate indicated that little or no carbonate ion could be removed by galena in the presence of dithiophosphate, although in its absence some carbonate abstraction (namely 140×10^{-6} mol per g on galena No. 4) was possible. These findings were qualitatively confirmed by the fact that X ray diffraction examination showed no carbonate present in the precipitate formed on adding lead ion to dithiophosphate solution buffered with carbonate. In the absence of dithiophosphate, basic lead carbonate was precipitated.

Rates: Curves of typical form for the rate of sorption of dithiophosphate from solution by ground galena are shown in fig. 2 for leached and unleached samples of preparation No. 5. Both curves show that there occurred a rapid initial uptake which soon terminated on unleached galena but which continued at a much reduced rate, sometimes negligible, on the leached. This secondary process was absent when the experiments were conducted in nitrogen instead of air. Many additional runs with other galena preparations and other solution concentrations showed that the rapid initial process was 90 to 100 pct complete in the first 10 to 12 min although the total quantity taken up in that period varied widely. It was clearly evident that, for the times usually involved in flotation, the fast initial process was the one of interest. Evident also was the fact that a large amount of agent was consumed by the leachable coating of galena.

Isotherms: The dependence of the amount of dithiophosphate taken up on the concentration of the solution is shown in fig. 3 for unleached and leached mineral, again from batch No. 5. Since these data were taken for a 12-min period, on the basis of the previous rate studies they are concerned only

Table II. Calculated Constants for the Isotherms

Galena, No.	Maximum Uptake, Mol per g $\times 10^6$	a Ce per g	n
4	800	2.06	0.81
5 (unleached)	560	1.21	0.79
6 (leached)	250	1.25	0.89
2	(70)[a]	0.57	0.92
Single crystal	(50)[b]	0.012[c]	0.70

[a] The highest value observed; the isotherm was not carried to the point where it leveled off.
[b] For 1000 cm² per g on the basis of 0.05×10^{-6} mol per cm² at a concentration of 100×10^{-6} mol per cc.
[c] On the basis of 1000 cm² per g.

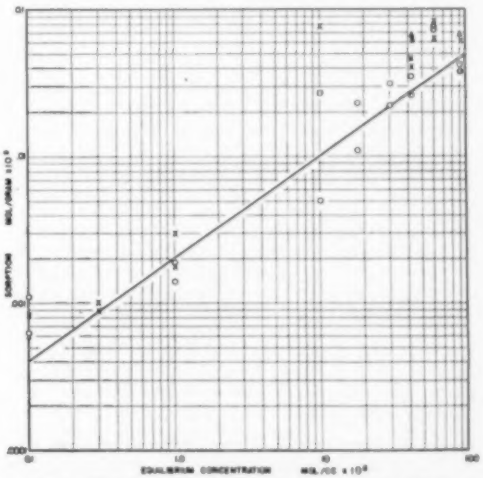

Fig. 4—Uptake of dithiophosphate on single galena crystals.

10 min sorption time. Temperature 25°C.

Table III. Desorption of Dithiophosphate from Unleached Galena No. 4

Time of uptake, 12 min
Time of desorption, 20 min
Desorption into 20×10^{-5} mol per cc Na_2CO_3

Sorption		Desorption	
Solution Conc., Mol per Ce $\times 10^6$	Dithiophos. Taken Up, Mol per G $\times 10^5$	Dithiophos. Desorbed, Mol per G $\times 10^5$	Conc. of Resulting Solution, Mol per Ce $\times 10^6$
3.2	42	19	0.8
12.1	170	71	3.0
30.5	468	101	4.3
46.9	613	90	3.8
56.4	804	85	3.6

Table IV. Desorption of Dithiophosphate from Single Crystals

Time of uptake, 10 min
Desorption into H_2O

Sorption		Desorption	
Solution Conc., Mol per Ce $\times 10^6$	Dithiophos. Taken Up, Mol per $Cm^2 \times 10^9$	Time, Min	Dithiophos. Left on Surface, Mol per $Cm^2 \times 10^9$
10	0.0083	7	0.0012
30	0.062	2	0.041
		7	0.029
		22	0.023
		16 hr	0.003

with the rapid initial process. The figure also serves to bring out clearly the larger uptake by the unleached mineral.

For all the galena preparations the isotherms were of the same type but the magnitude of the uptake was variable. They were conventional in shape, rising initially as the concentration increased but eventually leveling off. Although some could be fitted to an equation of the Langmuir type, it was found that in general the initial portion, up to 20 to 40 $\times$ 10^{-6} mol per cc equilibrium concentration, gave reasonably straight lines on a log-log plot; above this concentration region, the amount taken up was practically independent of concentration.

The fact that the initial portion fitted a log-log plot is expressed by the usual Freundlich equation*:

$$M = aC^n$$

where M is the mols taken up per gram, C is the concentration, and a and n are constants. It appeared that the value of n was sensibly constant ($\pm$ 10 pct) for all galena samples but factor a was altered; table II gives values which were calculated. The constancy of n indicated that the curves were all essentially of the same form, which suggested that although the preparations were quantitatively different they were qualitatively the same.

Single Crystals: Experiments of the type described above show in an indirect way the quantity of agent taken up by the mineral. It was one of the advantages of the radiotracer method that its sensitivity permitted direct measurement of the amount of agent on the surface of a single galena crystal 0.2 to 1 cm^2 in nominal area. For this purpose, freshly-cleaved, single galena crystals were employed, and the results are shown in fig. 4 on a log-log plot of the amount of dithiophosphate (ammonium diisopropyl) taken up per cm^2 versus the concentration of the solution (unbuffered).* It is noteworthy that a thousandfold concentration range was covered, and that definite amounts of dithiophosphate were taken up by the galena even from solutions of extremely low concentration. The results scatter very considerably, not even the complementary faces of a given cleaved crystal agreeing in many instances. This scatter may have been due in part to uncertainties in the area counted and to cracks, and so on, although it is thought that the macro area represented the true area within a factor of two. In general, however, the scatter was correlated with the presence of stained areas on the freshly cleaved faces, for high radioactive counts were always associated with visible stains. In spite of the scatter, it is evident that the data here, as with the ground mineral, can be approximately represented on the log-log plot by a straight line. Of considerable interest is the fact that here also the value of the exponent, n, approximates that obtained in the case of the ground minerals, as is shown in table II, although the value of a is again different.

* Mention may be made here that a few similar experiments were carried out on single crystals of sphalerite. Within the limit of error no agent was consumed unless the mineral had been first activated in copper sulphate. Although this was as expected, the radiodithiophosphate used was unfortunately of low specific activity so that small amounts which might have been taken up by the unactivated sphalerite would not have been detected.

Desorption: In addition to the above studies on the uptake of dithiophosphate, an investigation was made of the desorption of the consumed dithiophosphate from the galena, both for ground galena and for single crystals. Here again, it may be emphasized that without the radioactivity method, experiments such as these could scarcely have been accomplished. Results are recorded in table III which includes the conditions under which the uptake was effected as well as the amount desorbed and the concentration of the resulting solution. Except for the one instance where the starting amount was small, it is evident that the quantity desorbed was independent of the amount initially on the surface. Time of sorption, up to 60 min at least, did not alter this result. Whether or not greater desorption would have occurred had the time been extended is uncertain; experiments on another galena showed approximately the same amount given up in 3 hr as in 20 min. In the latter case, however, the final concentration of the desorption solution was about 16 $\times$ 10^{-6} mol per cc, about 4 or 5 times larger than obtained in the trials of table III.

Typical desorption results from single crystals are given in table IV. They show that the agent was partially desorbed from the surface in the time permitted. Qualitatively, therefore, these experiments are in line with the results with ground mineral.

Table V. Exchange of Dithiophosphate between Solution and Sorbed Phase

Galena No. 4
(20×10^{-5} mol per cc Na_2CO_3)

Dithiophos. Conc., Mol per Ce $\times 10^6$	Total Mols Present, $\times 10^6$		Exchange Time, Min	Fraction Exchanged
	In Solution	On Galena		
17.0	339	674	2	0.66
7.2	143	410	2	0.67
4.5	90	189	2	0.78
4.3	86	196	10	0.92

Exchange: The exchange of dithiophosphate ion in solution with radiodithiophosphate that had been taken up by galena was examined to provide further information on the nature of the interaction product. This is a type of experiment which is only possible by tracer techniques. Table V records some typical data on the "fraction exchanged" (ratio of observed count to count expected at statistical equilibrium). It is evident that exchange was not instantaneous but required 10 min or more for completion. None the less it is clear that a dynamic exchange of ions between solution and sorbed phase was occurring during times of the same order of magnitude as are involved in the rapid initial sorption process and in flotation. This general finding was confirmed by reverse exchange experiments, in which inactive di-

Table VI. Exchange of Dithiophosphate with Lead Dithiophosphate

Dithiophos. Conc., Mol per Cc × 10^5	Total Mols Present, × 10^6		Exchange Time, Min	Fraction Exchanged
	In Solution	In Precip.		
5.4	107	468	10	0.24
5.4	107	221	10	0.33
5.4	107	103	10	0.20
5.3	80	20	10	0.25

Table VII. Effect on Dithiophosphate Concentration on Flotation

(20×10^{-5} mol per cc Na_2CO_3)

Equil. Conc., Mol per Cc × 10^5	Dithiophos. Taken Up, Mol per G × 10^5	Flotation Index
Galena No. 2		
0.16	0.5	0.7
0.35	0.7	0.9
Galena No. 5 (leached)		
0.64	4.2	0.7
1.4	8.0	0.9

thiophosphate was present on the galena and radiodithiophosphate ions were in the solution.

In order to view the above results in better perspective, similar exchange experiments were carried out with freshly-precipitated, lead radiodithiophosphate and dithiophosphate ion; table VI shows representative results. The data were necessarily obtained at higher total mols in the precipitate than existed on the galena but the ratios of the mols in solution to those in the precipitate approximated those of table V. It is evident that appreciable exchange did occur between freshly-precipitated, lead dithiophosphate and dithiophosphate ion in solution. Although it would appear that the rate was somewhat slower, it is difficult to make a direct comparison of the rates since the extrapolation of the findings here at high concentration to those at the lower concentrations prevailing in the experiment with galena is made uncertain by the complexity of heterogeneous exchanges of this type.[7]

Flotation: Variations in flotability of the different mineral preparations were found, although the procedure employed gave reproducible values of flotation index for a given batch. The notable point was that nearly complete flotation was accomplished at low solution concentration and at low amounts of agent taken up. Table VII illustrates dithiophosphate quantities found necessary to produce flotation index values of 0.7 and 0.9 for two galena preparations.

Discussion of Results

Numerous investigations attest to the chemical heterogeneity of the galena surface.[6] Either in air or in solution galena can oxidize to compounds of the general type PbS_xO_y[6]. In support of this, electron diffraction patterns made during this investigation on weathered single crystals identified the surface product as $PbSO_4$. Very probably a galena surface can vary from a monolayer of oxygen all the way to several layers of lead sulphate. The results presented above on the interaction of galena with dithiophosphate seemed in accord with this picture, for they were strongly dependent upon the state of the galena surface. Variable uptake of dithiophosphate with weathering, the greater sorption of unleached over leached mineral, and the increase in uptake by stain on single crystals all point in this direction. It is most probable that an oxidized surface was always involved, even with leached galena or with freshly-cleaved single crystals, and that at no time was a clean, lead-sulphide surface at hand.

Table VIII. Estimated Limiting Concentrations

Galena	Pb Available (Assumed), Mol × 10^6	Initial Carbonate, Mol × 10^5	Estimated Limiting Conc., Mol per Cc × 10^5	Lowest Conc. Investigated, Mol per Cc × 10^5
Single crystals	25	0	0.8	0.1
Galena No. 2	160	400	5	1.5
Galena No. 4	430	400	0.5	
Galena No. 5	300	400	0.7	

As mentioned previously, however, similarity in form of the isotherms for all the galena preparations, regardless of the differences in the amounts taken up, suggests that the heterogeneity was one of degree rather than of kind. Finally, the general shape of the isotherm may be interpreted as arising from interaction with a number of surfaces, each of slightly different properties and each present in varying amounts from preparation to preparation. In view of this surface heterogeneity, it is evidently very unlikely that the results can be satisfactorily represented by any one theory. The considerations made in the following are confined to the rapid initial interaction completed in 10 to 12 min.

The observed uptake of dithiophosphate by galena was not a physical adsorption on the galena surface resulting from van der Waal's forces or like causes. Such physical adsorption is in general characterized by very rapid rates of sorption and desorption, and by practically instantaneous exchange between the adsorbed and the solution phases. None of these characteristics were observed here. On the contrary, the rate of uptake, although fairly rapid, was finite and measurable, the desorption was slow and incomplete, and the exchange required a time comparable to the sorption time. Accordingly the interaction was either some kind of a chemisorption or chemical reaction at the surface.

Formal distinction between chemisorption and surface chemical reaction is difficult since the interaction forces of chemisorption are of the same type as are concerned in chemical reaction. Both require activation energies and proceed at finite rates. Chemisorption is usually considered to involve no more than a monolayer, and the term will be used in this sense here. On the other hand, a sorptive process will be defined as one of chemical reaction when an isotherm of the form required by the phase rule is obtained, namely no sorption occurring below that limiting concentration fixed by the normal solubility product of the compound formed.

Assuming that lead dithiophosphate was the only product formed as a result of a possible chemical reaction between dithiophosphate and galena, an attempt was made to apply solubility product considerations.* The difficulty in such an application here lay in the fact that the amount of lead available for reaction was uncertain. Table II suggests that since the total amount of dithiophosphate taken up by the various galenas was 50 to 800 × 10^{-6} mol per g, 25 to 400 × 10^{-6} mols of lead was available, in 20 cc of solution. Using these figures as representative of the maximum possible lead-ion concentration, the limiting dithiophosphate-ion concentration fixed by the solubility product was calculated; the assump-

* R. K. Madison of the Laboratories, using radiolead as an indicator, found the solubility of lead sec.-butyl dithiophosphate was 0.37 × 10^{-5} mol per liter of water, pH 6 at 25°C. The solubility determination using radioactive dithiophosphate as indicator gave a figure of 0.53 × 10^{-5} mol per liter, in satisfactory agreement. The solubility product calculated therefrom is 0.37 × 10^{-15} (mol per liter)[3]. In carbonate solution, 20 × 10^{-5} mol per liter, a concentration of dithiophosphate ion of 7 × 10^{-5} mol per liter was observed. This is almost exactly the value calculated from the observed solubility of lead dithiophosphate in water and the known solubility product of lead carbonate, 0.16 × 10^{-13} (mol per liter)[2],[3].

tion was made on the one hand that no carbonate formed and on the other that lead carbonate and lead dithiophosphate co-precipitated. The results are shown in table VIII. Unfortunately, only with galena No. 2 and with single crystals were measurements made at low enough concentrations to permit comparison between calculation and experiment. Although definite conclusions cannot be drawn in view of the uncertainties of the calculations, the fact that uptake was observed below the estimated limiting concentration suggests that the formation of lead dithiophosphate was at least not the sole process. This is supported by the fact that the sorption isotherms were not of the form defined for simple reaction. The flotation curve of fig. 1 supplies additional evidence by showing that measurable flotation, and therefore sorption, took place at concentrations as low as 0.1×10^{-4} mol per cc. From these considerations, it does not appear that only lead dithiophosphate was produced by the interaction of dithiophosphate with galena during the initial 12-min period.

Chemical reaction, as defined, cannot be entirely excluded, however, for the following shows that chemisorption cannot be the sole process occurring. Using 70 A^2 as the projected area of a di-sec-butyl dithiophosphate ion, as indicated by Fischer-Hirschfelder models, a monolayer would be equivalent to 0.025×10^{-6} mol per cm^2 or 25×10^{-6} mol per g for a surface of 1000 cm^2 per g. Table II indicates that amounts much larger than required for a monolayer were taken up, and consequently positive support is lacking for a theory that a monolayer formed. Further, the desorption data suggested that a product had formed which, in 20×10^{-6} Na_2CO_3, was in equilibrium with a dithiophosphate-ion concentration of 4 to 16×10^{-6} mol per cc. The equilibrated value for lead dithiophosphate was 7×10^{-6} mol per cc in carbonate solution. This may indicate that lead dithiophosphate existed after 20 min, but the data cannot establish its formation as the sole primary product in the interaction.

It was not possible from the exchange rates to clarify further the nature of interaction. If exchange had occurred with the sorbed phase on the mineral and not with the lead dithiophosphate, then the obvious conclusion could be drawn that the interaction did not form the lead dithiophosphate. Since appreciable exchange occurred in both cases no clear conclusion was possible.

From the foregoing discussion, the interaction of dithiophosphate with galena appears to have characteristics of both chemisorption and chemical reaction. Its complexity probably lies in the heterogeneous nature of the galena surface. This may be visualized by supposing that the overall interaction is a composite of separate interactions of dithiophosphate with lead sulphide and all its oxidized forms. Each interaction must be controlled by a definite activation energy for a process having a definite, free-energy change. The surface at the end of the rapid initial sorption process is thus also complex and heterogeneous. Therefore, no single mechanism will account for the observed overall interaction. In this system, at least, these findings appear to be at variance with the generalized theory of chemical reaction[10] or for that of adsorption[7] for the interaction in mineral-collector systems.

The relation of the interaction to the flotation results is of interest. The speed of the sorption process appeared to be fast enough so that, in flotation times normally employed, a significant amount of sorption could occur. The process was quite irreversible. It was also brought out that flotation was achieved at relatively low concentrations of dithiophosphate and relatively small amounts taken up. These latter were lower than estimated for a monolayer on the nominal galena surface. Evidently, therefore, if the particles of galena were sufficiently small, only a partial covering was necessary to impart to them a sufficiently hydrophobic character.[12] The fact that these low concentrations were operative indicated that the interaction products adhered to the galena surface. Even though part of the weathered coating could be leached, apparently this did not result in a loss of the dithiophosphate for flotation.

Summary

Radiotracers were demonstrated to be of considerable value in the study of a typical mineral-collector system, dithiophosphate-galena. Since they provide a rapid and sensitive analytical method, they are especially applicable to the conditions of flotation.

For the chosen system, measurements were made of the rate of uptake, the isotherms, the desorption, and the exchange of dithiophosphate on both ground mineral and single crystals.

It was concluded that no single mechanism could account for the observed overall interaction. The process was complex because of the heterogenous nature of the galena surface. The interaction was not one of physical adsorption, but appeared to have characteristics of both chemisorption and surface chemical reaction.

It was shown that sorption proceeded fast enough so that significant uptake of dithiophosphate occurred within times normally employed in flotation. The process was quite irreversible. Flotation was achieved at relatively low concentrations of agent and at relatively small amounts sorbed.

Acknowledgment

We wish to express our appreciation for encouragement and support from S. J. Swainson and R. B. Barnes, for helpful discussion with R. B. Booth, S. A. Falconer, J. E. Carpenter, and for technical assistance from various other members of these Laboratories during this investigation.

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Experiences with a Density Recording and Controlling Instrument for Heavy-media Separation Units

Although determining and controlling specific gravity of operating medium in a heavy-media plant manually presents no problem, there are advantages to automatic recording and control. The two installations discussed employ the usual method of measurement. Pneumatic control with band control and reset features is used to regulate dilution water and hence specific gravity. Difficulties have been mechanical in nature and associated with obtaining a proper sample and maintaining tight air connections in bubble-tube piping.

by James J. Bean

THE task of measuring the specific gravity of the operating medium in a heavy-media separation system has never presented a particularly difficult problem because the medium is fairly stable and the overflow of the separatory vessel, as well as its underflow, can be sampled easily and accurately and the specific gravity of the suspension determined easily by weighing a known volume. However, while this method is simple and accurate it does require the operator to take the sample by hand and to weigh it and there is considerable temptation to avoid the periodic sampling if everything seems to be going well, or if something is occupying the attention of the operator. Furthermore all operators do not sample in exactly the same manner and considerable practice is required for two operators to be able to "check" each other to the last few hundredths, particularly if the sample is cut underneath the drainage screen where location of the point of sampling and load on the screens tends to influence the determination.

While none of the above presents much of a problem, we have all recognized that some mechanical method of continuous measurement and recording would be advantageous since the operator would merely have to glance at the meter to check the gravity and to have an indication of the trend of any changes. Also if the instrument were of the recording type, a permanent record would be available for the guidance of the superintendent.

The Eagle-Picher Mining and Smelting Co. was the first heavy-media user to actually install such a recording meter. In 1946 they installed in their Central Mill at Cardin, Okla., a specific gravity recorder manufactured by the Bristol Co. of Waterbury, Conn. R. A. Barnes, of the Bristol Co., working with E. H. Crabtree, Jr. and Elmer Isern, of Eagle-Picher, made the application and worked out the problems of sampling and measuring. Their attempts to measure the specific gravity of the medium in the cone itself were not entirely successful and they resorted to an outside sample tube for actually making the determination. Because of the particular flowsheet used, it was possible to tap off from the medium return pipeline a stream of medium and divert it into the sampling tube, which was provided with a constant level overflow and a spigot underflow, and into which the bubbler tubes dipped.

JAMES J. BEAN, Member AIME, is Mineral Dressing Engineer, American Cyanamid Co., Stamford, Conn.

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The Eagle-Picher installation was successful and its possibilities were recognized by the Mineral Dressing Laboratory of the American Cyanamid Co. It was decided to install a similar unit in the heavy-media pilot plant to investigate further its possibilities. Chief among these was the continuous record which it was felt would be proof of the steadiness of the gravity in a heavy-media cone, something which is not always appreciated by potential users.

Because the heavy-media pilot plant is required to operate at a wide range of specific gravities, it was realized that the unit would have to record all gravities from 1.25 to 3.50, and do it to the nearest 0.01. It would not be necessary to record all of this wide range on a single chart and the method selected was to have 4 bands, each band range overlapping the other a small amount and calibrated so that with standard charts one division would represent 0.01 sp gr. A shift from one band to another could be arranged without alteration of the instrument itself, being accomplished by a simple change in the bubble-tube lengths, as described later. Accordingly, a recording type instrument was purchased and installed.

Because there were some advantages in doing so, the first installation attempted to measure the gravity of the cone proper by placing the bubble tubes in the cone. This was not at all satisfactory and the second scheme utilized a fixed vertical screen at the surface of the cone, and an external sample-tube arrangement. We were particularly anxious to make this work as we felt it would be advantageous to measure the top level of medium where the separation was actually being made, but we were doomed to disappointment because it was impossible to keep the screen clean of float.

Since the top gravity of the cone is the most convenient place to sample for control, a launder about 2 in. wide was installed longitudinally beneath the

center of the float drainage screen. This sloping launder took a continuous sample of the float medium draining through the float screen, and diverted it into the sample tube. Following its installation no further difficulty was experienced in securing continuously a representative sample of float drainage medium.

When the problem of sampling was overcome, we immediately started to get recordings on the meter, but because of a variety of causes, it was some time before they were accurate and dependable. These difficulties are described later in this paper.

When it appeared that recording the density was going to be satisfactory even though there were some "bugs," the possibility of not only recording the density but actually controlling it was discussed. It was thought that such control was possible if the specific gravity of the medium could be adjusted by controlling one element such as the water or the densified medium.

Because the densified medium returning to the circuit is always at higher gravity than the circulating medium, the controller could regulate either the amount of densified medium returned to the circuit, or could regulate the amount of water being added to the circulating medium. The latter method was chosen because of its simplicity and furthermore because experience had indicated that this was the best method for manual control.

Description of Control Method

The principle involved in mechanically measuring specific gravity is an old one, and is employed by most, if not all, of the manufacturers of specific-gravity recorders. It utilizes two pipes of different lengths which dip into the medium or liquid to be measured. These pipes are supplied with a small amount of air through sight bubblers. The pressure required to overcome the head of medium or liquid is of course different for each pipe, and builds up until the air bubbles out of the bubble pipe. The air lines supplying the bubble pipes are connected, one to each side of a differential manometer, and thus, the differential in air pressure in the two bubble pipes is measured. This differential is recorded as specific gravity since obviously any change in specific gravity of the medium will be reflected immediately as a corresponding change in the air pressure. The two tubes of different lengths are employed to eliminate the need of an absolutely uniform overflow level in the sample tube. Such a uniform overflow level is virtually impossible to obtain, although it has been found necessary to maintain it as closely as is practical for close indication.

As mentioned earlier, the instrument installed in the heavy-media pilot plant had to be able to record a wide range of specific gravities. The change in range is easily accomplished by changing the difference in orifice submergence between the two bubble pipes, the difference being less for the higher gravities. Actually one set of bubble pipes is used with the long one provided with tapped holes at the proper points, which are normally closed with plugs. By unplugging the proper hole, the range of the instrument is easily set for the particular medium gravities that are to be measured.

The actual specific-gravity ranges, together with the corresponding difference in orifice submergence (dimension *H*, fig. 1) of the bubble pipes, are as follows:

"H" in In.	Specific-gravity Range
20	1.25 - 1.75
16 2/3	1.50 - 2.10
12 1/2	2.00 - 2.80
10	2.50 - 3.50

The arrangement of the equipment is shown in fig. 1, in which (*1*) is the sample launder under the float drainage screen. (*2*) is the sample tube with its constant level overflow, (*3*), discharging back into the medium drainage sump (*4*), and its underflow spigot (*5*) discharging into a box and thence back into the medium drainage sump. (*6*) are the bubble pipes with a difference in submergence *H*

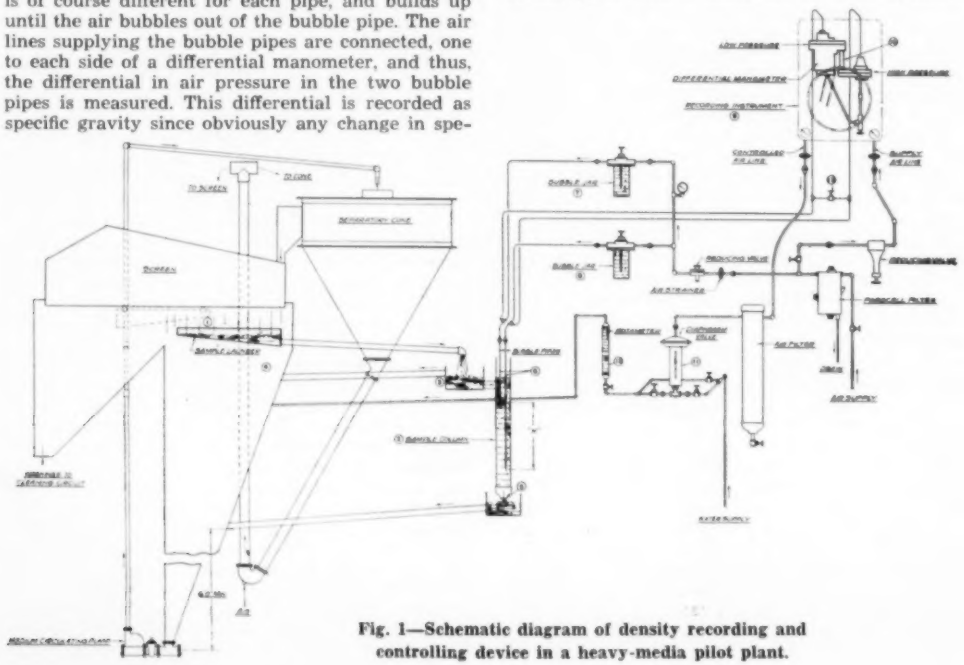

Fig. 1—Schematic diagram of density recording and controlling device in a heavy-media pilot plant.

which as explained above is adjustable for different gravity ranges. Air is supplied to the bubble pipes through the bubbler jars (*7*) and (*8*) which have needle valves for adjustment. The actual measurement and record is made by the differential manometer (*9*).

The differential manometer has an internal construction as shown in fig. 2. From this drawing it is apparent that the instrument is a U tube filled with mercury. The low-pressure side is provided with a float which operates the recording pen through a suitable packing gland. The high-pressure side is simply a reservoir of mercury. Both chambers are provided with check valves to prevent the loss of mercury. At the bottom of the U tube a needle valve is provided to damp the pen movement if necessary, but our experience has been that it is not necessary to damp the pen. Because it is desirable to have a relatively short range of specific gravity over which the meter measures, a counterbalance is built into the meter body. This counterbalance suppresses the zero point, that is, it is built in such a way that the two mercury columns of the U tube are not level until the minimum reading is reached. By this arrangement the full pen movement can be confined to an 0.5 difference in specific gravity between low and high gravity. The counterbalance feature is provided by raising the low-pressure side of the meter above the high-pressure side by means of a spacer (*10*), fig. 1, and a longer U tube. By raising this chamber it can be seen that considerable pressure must be placed on the high-pressure chamber before mercury is forced into the low-pressure chamber and the float moved.

Once the gravity has been measured, controlling equipment can provide a variable air pressure proportional to the displacement of the pen from a given set point. This controlled air supply can in turn operate a diaphragm valve, or some positioning device. Thus, if the gravity of the medium in the cone circuit can be controlled by varying one element, the instrument can control the gravity of the medium. As it happens the heavy-media separation system is generally so operated, that sufficient medium at a high enough gravity is returned from the densifier to the medium drainage sump to make necessary the continuous addition of a small amount of water in order to maintain the circulating medium at the proper gravity. Under manual control this water supply is varied by the operator and thus it was only natural that when the gravity recording meter started to function, the possibility of controlling this water by the meter was considered.

An instrument was purchased that was equipped with the necessary additional auxiliary units to control the water through a diaphram valve. The control unit is of the "Free Vane" type and is provided with "Reset" and "Proportional Band" features. The description of the method whereby these two features are obtained would be more involved than the scope of this paper provides, and we shall confine ourselves to a short description of what "Reset" and "Proportional Band" features do.

The proportional-band feature simply narrows the range through which the pen must move in order for the control air pressure to vary from 0 to 17 lb, which is the full pressure required to operate the diaphram valve. Perhaps this is more clearly stated by saying that a 100 pct proportional band would be the full width of the chart, that is to say, the pen would have to move from the lowest point

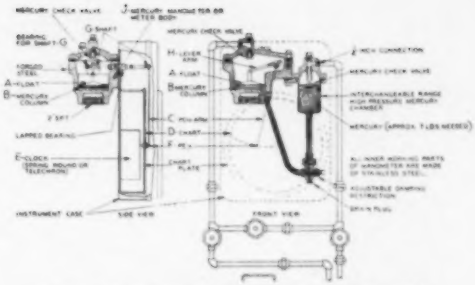

Fig. 2—Construction of differential manometer.

to the highest point of the chart in order to open or close completely the diaphragm valve on the water supply. Through proper linkage this percentage can be varied from 0 to 100. At 0 the controller would be an "on" and "off" affair since any change of the pen away from the set point would fully open or close the valve.

The reset feature functions only when the pen is off the set point when it operates in such a way as to tend to restore the pen to its set point. It moves at intervals which are adjustable through a needle valve, and by proper adjustment tends to eliminate the "offset" or variation between the set point and the actual control point, the latter being the point at which the system is stable for any particular valve setting due to the proportional-band control.

By properly adjusting both controls so that one supplements the other, it has been possible to control the water to the medium drainage sump so accurately that the pen recording the specific gravity virtually draws a circle on the chart, fig. 3.

The controller operates in such a way that the controlled air pressure increases as the specific gravity decreases, and an increase in control air pressure tends to shut the diaphragm valve, (*11*) fig. 1, which supplies water to the medium drainage sump and thus acts to increase the specific gravity of the circulating medium. In the pilot plant, the actual amount of water being supplied is measured with a rotameter (*12*) but this refinement would not be necessary in an operating plant. Also because we erred in estimating the flow through the diaphragm valve, it is somewhat too small to supply the full water requirement and an auxiliary supply, manually controlled, has been provided. There is considerable debate as to the merits of this arrangement over a single diaphragm valve, and we intend to install a large valve with adequate capacity to settle this point.

Although the units we have installed are standard units, adapted to the particular job to be done, due to the fact that the continuous measurement of specific gravity is relatively new, particularly in the case of suspensions, there are some features which are novel to this installation.

As mentioned earlier, the matter of securing a representative sample for measurement and control purposes has been something of a problem, and while it is believed that the sample launder arrangement is entirely adequate, it does have the disadvantage of requiring headroom where the latter may be at a premium. Some other arrangement may ultimately be devised, which would still allow the measurement of the gravity of the overflow medium, and require less headroom.

We suspect that the bubble pipes may accumulate

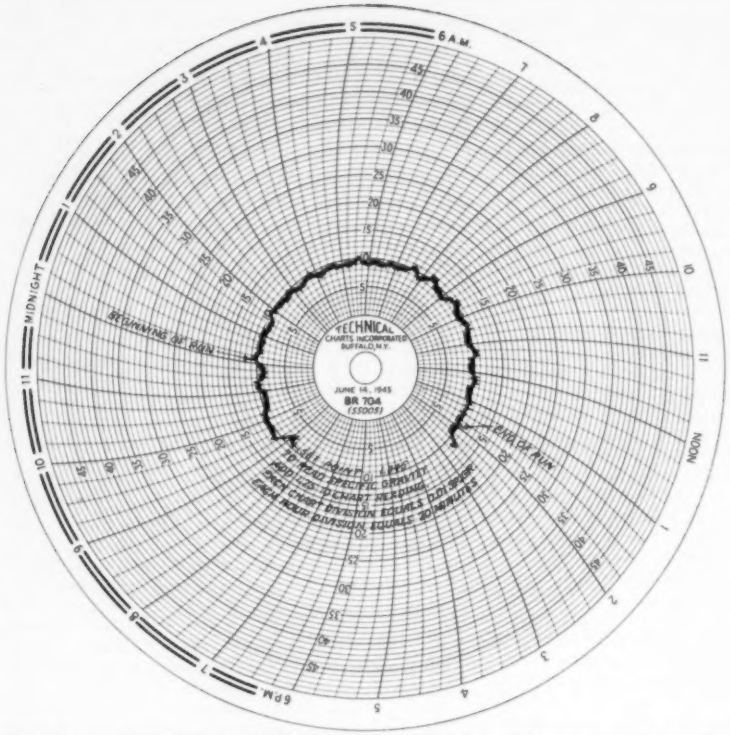

Fig. 3—Chart of specific gravity control in heavy-media pilot plant.

Coal plant treating coal using magnetite medium. Overflow gravity = 1.345.

medium on themselves, which might cause the opening to close after a period of operation. A preventitive measure would seem to be a fixed schedule of wash down to remove such accumulations. Other measures suggest themselves such as feeding water down the bubble tube, but because of the short runs made at the pilot plant, we do not get the accumulations and so cannot work out the answer.

The greatest difficulty we have experienced has been in getting absolutely air tight connections in the bubble-pipe system. Originally ⅛-in. iron pipe was used, but the system leaked so badly that it was removed after only a few days of trial. The next attempt was made with ¼-in. copper tubing having compression fittings but this also leaked and was replaced with sweat-solder fittings.

As originally installed, each line from the bubble pipes to the manometer was equipped with a valve for test purposes. These valves leaked and were therefore removed. Only the balancing valve, (*13*) fig. 1, has been retained, and in this position gate valves, globe valves, and needle valves were investigated, and at time of writing a diaphragm sealed type of globe valve was being tried. This diaphragm sealed valve has a packing gland and seems to be air tight.

The difficulty of making pipe connections air-tight to small flows of air at about 10 psi has been incredible to us. Of course with a bubble rate of about 1 per sec, the loss of one or two bubbles a minute is serious, and the pipe joints must be absolutely leak-proof. Joints that will not show a leak to soap suds will still leak too much for accurate recording and the only way to detect them is to isolate a section of the system and use the bubbler itself as the detector. If it stops bubbling eventually with air pressure on its needle valve, the system is tight enough, though sometimes three quarters of an hour is necessary to bring the system up to pressure and to stop the bubbler.

However, copper tubing with sweated joints is relatively easy to install air tight, and if it is used there should be no difficulty in getting good air tight joints.

When the controller was installed some difficulty was experienced in getting the right setting for the proportional band and the reset, largely because neither ourselves nor the manufacturers had had any experience with such an installation. This inexperience coupled with leaking tubes made things interesting for some time, but now we believe we can diagnose most of the usual faults in the system and correct them without too much difficulty.

Although we have a Bristol instrument in the pilot plant, a number of manufacturers are able to supply similar equipment and have done so. Among these manufacturers are Moore Products Co., the Brown Instrument Co., and the Foxboro Co. No doubt others can supply similar equipment because it is essentially the differential manometer used in flow meters of the orifice type, and the usual control equipment used with such meters.

Acknowledgment

Grateful acknowledgment is made to Messrs. Hamilton Bristol, G. M. Heslin, and W. C. Juram, of the Bristol Co., for their determined efforts to make the installation a success, to Mr. Pickens, of the heavy-media pilot plant, who had to live with the unit after it was installed and to the officials of the American Cyanamid Co. for permission to publish this paper.

Behavior of

Mineral Particles in Electrostatic Separation

by Shiou-Chuan Sun, J. D. Morgan, Jr., and R. F. Wesner

A new apparatus termed a "distribution analyzer" is introduced to ascertain the optimum conditions of electrostatic separation of minerals. It was found that with similar surface condition, testing environment and a constant feed rate of not higher than one layer of particles fed on the grounded roll of the separator, the behavior of each component mineral in a synthetic mixture or ore is roughly similar to that of the same mineral when tested alone.

FOR successful electrostatic separation, the behavior of the component minerals of the tested ore under various conditions should be known. In the course of investigation, it was found that these are difficult to measure accurately by the presently available methods. The method used by Johnson[1] is handicapped by the fact that particles in the extreme front and rear end of the falling stream are difficult to control or observe accurately. Furthermore, this method is not applicable when the tested sample consists of more than one mineral. The deflection method, as reported by Fraas,[2] is inadequate to show the differential amounts of mineral particles deposited at the various horizontal positions under the grounded roll of the separator. This weakness is further noted when the tested sample contains more than one mineral.

SHIOU-CHUAN SUN, Member AIME, is Assistant Professor of Mineral Preparation, Pennsylvania State College, State College, Pa.; J. D. MORGAN, JR., is Assistant for Materials and Stockpile Policies, National Securities Resources Board, Washington, D. C.; and R. F. WESNER, Junior Member AIME, is Engineer, McNally Pittsburg Manufacturing Corp., Pittsburg, Kans.

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The purpose of this paper is to introduce a new method capable of measuring the behavior of mineral particles under various conditions, and of defining the optimum condition for separation. The main factors which influence the behavior of mineral particles in electrostatic separation are also described.

Experimental: A new apparatus termed, "distribution analyzer," was used to replace both the dividing gauge and the two attached collecting chutes of a Johnson separator.[3] This distribution analyzer consists of sixty or more 1.0-cm wide cells as shown in part in fig. 1; it has been fully described in a previous article.[4]

Except where otherwise stated, the testing procedure used in this paper is as follows: (*a*) calculate the weight of feed for a 1-min run at a constant feed rate of one layer of closely packed particles falling on the grounded roll; (*b*) weigh and dry the feed at 100°C for 12 hr; (*c*) place the

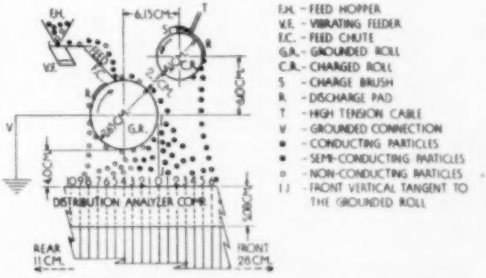

Fig. 1—Schematic diagram showing the operation of the electrostatic separator in combination with a distribution analyzer.

top of the distribution analyzer level 4 cm below the bottom of the grounded roll, so that the cells are longitudinally parallel to the grounded roll and the zero edge of a suitable cell is directly under the front vertical tangent to the grounded roll (see fig. 1); (*d*) with the charged roll at a negative potential of 15.5 kv, the dry hot sample is introduced into the feed chute at a constant feed rate; and (*e*)

finally the deposited mineral particles are removed from each cell and weighed and analyzed when the tested sample contains more than one mineral. Weighing alone is sufficient if the tested sample contains one mineral.

The weight of feed was calculated according to the following equation:

$$Q = dwvgcn \qquad [1]$$

in which Q is the weight of feed of sized mineral particles in grams for a minute run; d is the di-

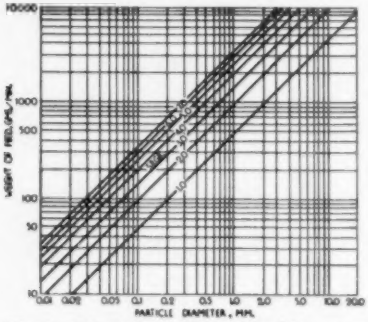

Fig. 2—Calculated weight of feed in grams for one minute runs at a feed rate of one layer of closely packed mineral particles fed on the grounded roll with assumed specific gravities from 1 to 7.

ameter of particle in centimeters; w is the effective width of grounded roll in centimeters, 11.35 cm for this separator; v is the linear velocity of grounded roll in centimeters per minute, 766.0 cm per min for this separator; g is the specific gravity of the tested mineral; c is the packing voids factor, equal to 0.524; and n is the predetermined number of layers of closely packed particles fed on the grounded roll. From Eq 1, the weight of feed in grams for minerals with assumed specific gravities from 1 to 7 are calculated and plotted in fig. 2. The curves in fig. 2 can be expressed mathematically as follows:

$$Q = kd^m$$
$$\log Q = \log k + m \log d \qquad [2]$$

where m is the slope and k is the intercept on the ordinate of 1.0 abscissa. To use the curves of fig. 2 for any roll-type separator, multiply the observed quantity from fig. 2 by a factor $wv/3694$. Symbols w and v are the effective width and linear velocity, respectively, of the grounded roll of the subject separator.

Mineral samples were prepared from selected lumps of high purity by crushing, grinding, and screening to the desired sizes. All the sized samples except coals were cleaned by magnetic separation and tabling, and finally washed with very dilute hydrochloric acid and distilled water. The cleaned samples were air dried and stored in glass jars.

Weight Per Cent Distribution: The weight per cent distribution curves of figs. 3 to 10 were obtained in the following manner: the weight per cent for a given cell is plotted at the midpoint of that cell and these points have been connected with solid and broken lines as shown. Where the points are connected by a solid straight line the following mathematical equation applies:

$$\text{Log } W = k + mx \qquad [3]$$

in which W is the weight per cent of particles deposited in any cell of the distribution analyzer, x is the horizontal distance between the central point of a cell and the front vertical tangent to the grounded roll (the zero distance on the abscissa), k is the ordinate intercept at zero abscissa, and m is the slope with minus and plus sign for the front and rear, respectively.

Eq 3 is not applicable to the broken line portions of the curves, which are believed to be caused by the combination of two interrupting forces; one, a mechanical rubbing of the attached particles on the rolls by the discharge pads, and, the other, an excessive attraction between mineral particles and charged roll. The former force may play a larger role than the latter, when nonconducting particles of medium size (48 to 150 mesh) are tested under moderate potentials (8 to 15 kv). The latter force may become the more dominant factor, when conducting particles of fine size are tested under high potentials.

Optimum Condition of Separation: There are two methods to ascertain the optimum condition of electrostatic separation of minerals. One is to test the ore sample directly under various conditions and is termed "ore-method." This can be exemplified by the tested data of a synthetic mixture of 82 pct anthracite coal, 14 pct quartz and 4 pct pyrite by weight, as shown in fig. 3. From curves 2, 4, and 6 of fig. 3, it can be seen that quartz and pyrite are partially removed from anthracite coal by

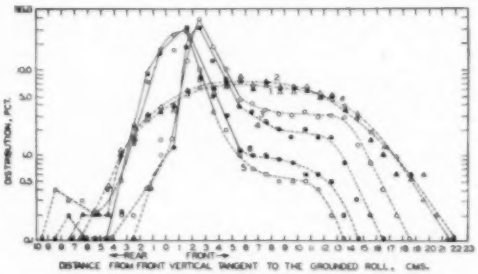

Fig. 3—The electrostatic behavior of 48x65 mesh mineral particles when tested alone and in a synthetic mixture.

Curves: *1*, anthracite coal; *2*, anthracite coal in mixture; *3*, pyrite preconditioned slightly with coal dust; *4*, pyrite in mixture; *5*, quartz preconditioned slightly with coal dust; and *6*, quartz in mixture.

placing the dividing gauge at 4 or 5 cm in front of the front vertical tangent of the grounded roll. The weight of the component minerals deposited in each cell of the analyzer was ascertained by means of sink-float determinations and by microscopic assay of the three fractions.

The other method is to test the individual component minerals instead of the ore itself under various conditions, and is termed "mineral-method." The results of the individual component minerals tested under the same environment are plotted in one figure and the optimum condition of separation can be determined by comparing the different figures. This can be visualized by comparing curves 2, 4, and 6, respectively, with curves 1, 3, and 5 of fig. 3. This method is based on the discovery that, under the same test condition and at a constant feed rate of not higher than one layer of particles fed on the grounded roll, the behavior of each component

Table I. Compiled Data Show the Relationship between Electrostatic Interfacial Resistivity and Electrical Resistivity, and also Dielectric Constant

Minerals	Chemical Composition	Electrostatic Interfacial Resistivity[1]	Electrical Resistivity[12,13,15]	Dielectric Constant,[12,14,15]	
		Relative Volt.	Ohm-Cm	e.s.u.	Frequency
		(A)	(B)	(C)	Cycles
Graphite	C	1.0–1.28	8×10^{-4}–0.5		
Anthracite coal	$C_{70}H_{41}O_2N$ (a)	1.28	70.0–5×10^{-6}	5.6–6.3	
Bitu. coal	$C_{135}H_{97}O_9NS$ (a)	1.45–2.23	(+) 10^{6}–10^{8}		
Mica schist	$(H,K)_2(Mg,Fe)_3Al_2(SiO_4)_3$	1.06–1.73	16—6×10^{4}	16.0–17.0	
Halite	NaCl	1.45	(+) 3×10^{8}–10^{12}	5.6–6.3	4×10^{6}
Bornite	Cu_3FeS_3	1.67	6×10^{-4}–0.5		
Chalcopyrite	$CuFeS_2$	1.67	0.1–10.0		
Fluorite	CaF_2	1.84	(+) 7.9×10^{7}	6.8	
Marcasite	FeS_2	1.93	$1.4\text{–}3\times10^{2}$		
Barite	$BaSO_4$	2.06		7.6–12.2	4×10^{6}
Serpentine	$H_4Mg_3Si_2O_9$	2.17	2×10^{4}–10^{8}		
Hematite	Fe_2O_3	2.23	3.5×10–1.2×10^{3}	25.0	
Talc	$H_2Mg_3(SiO_3)_4$	2.34		<10?	
Chalcocite	Cu_2S	2.34	$3.1\text{–}2.3\times10^{4}$		
Barium-strontium-titanate[16]		2.34(±)	(+) 10^{10}–10^{12}	6.5×10^{3}	
Pyrrhotite	FeS	2.34	5×10^{-4}–7.0		
Galena	PbS	2.45	3×10^{-4}–5.0		
Stibnite	Sb_2S_3	2.45	(+) 2×10^{6}–10^{14}		
Molybdenite	MoS_2	2.51	7.9×10^{-1}–1.6×10^{1}		
Ilmenite	$FeTiO_3$	2.51	2×10^{2}–5×10^{6}		
Siderite	$FeCO_3$	2.56	7×10^{8}	6.9–7.9	4×10^{6}
Tourmaline	$H_8Al_3(B.OH)_2Si_4O_{19}$	2.56		6.5–7.2	
Gypsum	$CaSO_4.2H_2O$	2.73		5.0–9.9	4×10^{6}
Magnetite	$FeOFe_2O_3$	2.78	*8×10^{-4}–5×10^{-1}		
Rutile	TiO_2	2.62–3.18		31–78.5	
Pyrite	FeS_2	2.78	*2.3×10^{-4}–9.2		
Dolomite	$CaMg(CO_3)_2$	2.95		7.3	
Sphalerite	ZnS	3.06	10^{6}–10^{9}	7.8–8.3	
Quartz	SiO_2	3.17–5.3	10^{7}–10^{14}	4.6–5.1	4×10^{6}
Quartz, fused	SiO_2	3.17–5.3		3.5–3.6	
Sulphur	S	3.9	10^{14}–10^{17}	3.6–4.7	4×10^{6}
Calcite	$CaCO_3$	3.9	5.5×10^{14}	7.5–8.5	4×10^{6}
Zircon	$ZrSiO_4$	3.96		11.0–13.0	
Wulfenite	$PbMoO_4$	4.18		23.8	
Apatite	$(CaF)Ca_4(PO_4)_3$	4.18	$>5\times10^{7}$	8.5	
Smithsonite	$ZnCO_3$	4.45		9.3–9.4	
Topaz	$(AlF)_2SiO_4$	4.45		6.2–7.4	
Corundum	Al_2O_3	4.9		11.4–13.2	
Aragonite	$CaCO_3$	5.29	1.4×10^{14}	9.1	

(+) Indicates that the value is apparently too high.
* Indicates that the value is apparently too low.
(a) Indicates that the formulas are approximate and given by ref. 20.

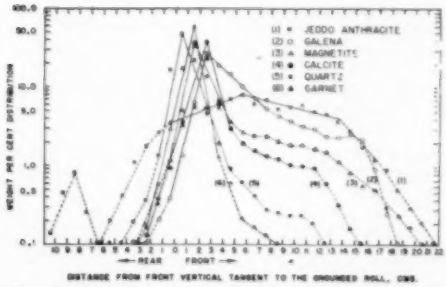

Fig. 4—The electrostatic behavior of some representative mineral particles of 48x65 mesh size.

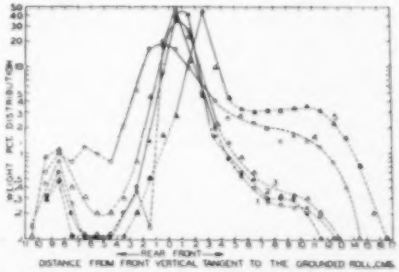

Fig. 5—Effect of chemical treatment on the electrostatic behavior of 48x65 mesh quartz particles.

mineral in a synthetic mixture or ore is roughly similar to that of the same mineral when tested alone.

It should be noted that the surface condition of the component minerals used for the mineral-method requires special attention. For example, it was found in the course of investigation that the conductivity of lower conducting minerals was somewhat increased by contamination with higher conducting minerals, such as coal, graphite, and galena. It should be noted that along this line the pyrite and quartz sample used, respectively, for curves 3 and 5 of fig. 3 were preconditioned slightly with a small amount of coal dust.

In comparison of the ore-method with the mineral-method, the former is more direct and accurate but requires analysis of the mineral particles deposited in each cell of the analyzer. The latter is indirect and less accurate but requires no analytical work; it is most convenient when the distribution curves of component minerals are available.

Surface Conductivity: The experimental data of fig. 4 show that, under similar test conditions, some minerals are repelled farther by the separator than others. This is due to the fact that minerals with a high-surface conductivity or low-surface resistivity are more easily polarized[3, 4] to the same polarity of the grounded roll than those of low-surface conductivity. The effect of the sign and magnitude of electrical charge on the behavior of mineral particles in electrostatic separation is given by Columb's law.

$$F = \frac{Q_1 Q_2}{kr} \qquad [3]$$

where F is the magnitude of the attractive or repulsive force in dyne between two charged bodies; Q_1 and Q_2 are, respectively, the charge of rolls and

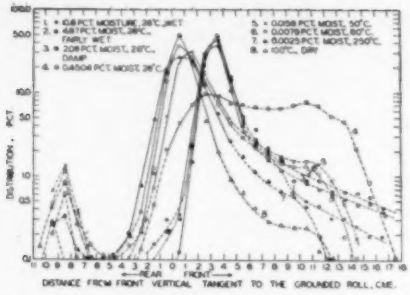
1. • 0.8 PCT. MOISTURE, 28°C., WET
2. ▲ 4.81 PCT. MOIST., 28°C., FAIRLY WET
3. ● 2.08 PCT. MOIST., 28°C., DAMP
4. ○ 0.4504 PCT. MOIST., 28°C.
5. ▽ 0.0058 PCT. MOIST., 50°C.
6. ◇ 0.0078 PCT. MOIST., 80°C.
7. ▼ 0.0025 PCT. MOIST., 230°C.
8. ◆ 0.0°C., DRY
DISTRIBUTION, PCT.
REAR
FRONT
DISTANCE FROM FRONT VERTICAL TANGENT TO THE GROUNDED ROLL, CM.

Fig. 6—Effect of moisture content on the behavior of 48x65 mesh quartz particles.

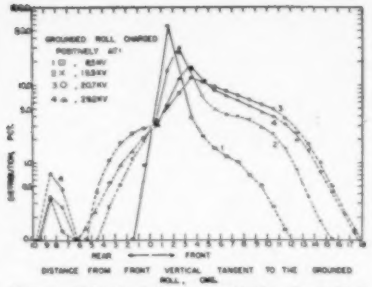
GROUNDED ROLL CHARGED POSITIVELY AT:
1 ○ , 85 KV
2 ▽ , 153 KV
3 ○ , 207 KV
4 ◇ , 250 KV
DISTRIBUTION, PCT.
REAR
FRONT
DISTANCE FROM FRONT VERTICAL TANGENT TO THE GROUNDED ROLL, CM.

Fig. 8—Effect of the magnitude of applied voltage on the behavior of 48x65 mesh calcite particles.

mineral particles in electrostatic units; k is the dielectric constant of the surrounding medium; and r is the distance in centimeters between the two charged bodies.

The behavior of mineral particles in an electrostatic field is a surface phenomenon and is governed chiefly by the surface atoms and ions instead of the internal composition of mineral particles. For example, the electrostatic behavior of quartz can be

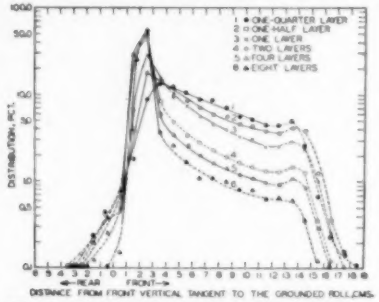

Fig. 7—Effect of feed rate on the behavior of 48x65 mesh galena particles.

altered by changing its surface condition with chemicals and moisture, as shown in figs. 5 and 6, respectively. The data of table I indicate that it is difficult to establish a relationship between electrostatic behavior (col *A*) and dielectric constant (col *C*) of minerals. This may be due to the fact that the dielectric constant[7-10] of minerals varies within narrow limits and represents more or less the electric conductance of the bulk body instead of the surface of the mineral particles. On the other hand, there is a general relationship between the established data of electrical resistivity (col *B*)[8, 10, 11] and the electrostatic behavior (col *A*) of minerals.

Chemical Treatment: The data of fig. 5 indicate generally that the electrostatic behavior of quartz particles is altered by chemical treatment, changing in the decreased order, lauryl amine hydrochloride, copper sulphate, hydrofluoric acid vapor, and barium chloride. This behavior depends chiefly on their surface conditions, which are in turn governed by the surface atoms and ions, as previously described. The surface condition and consequently the surface conductivity of quartz is altered by the absorption of aminium ions[13, 15] from lauryl amine hydrochloride solution, of copper ions[12] from copper sulphate solution, and of barium ions[14] from barium chloride solution. The effect of hydrofluoric acid vapor on the behavior of quartz may be caused by the formation and evaporation of silicon tetrafluoride vapor on quartz surface.

The curves of fig. 5 are: *1*, no chemical treatment; *2*, conditioned 5 min with 10 mg per liter of lauryl amine hydrochloride at 1.5 pulp dilution; *3*, conditioned 10 min with 50 mg per liter of $CuSO_4 \cdot 5H_2O$ at 3.0 pulp dilution; *4*, treated 15 min with hydrofluoric acid vapor; and *5*, boiled 15 min with 50 mg per liter of barium chloride at 1.5 pulp dilution.

Moisture Content: From curves 4 to 8 of fig. 6 it can be seen that to a certain limit, the magnitude of repelling of quartz particles by the separator is increased with the increase of moisture content. This phenomena can be explained by the established data[15, 16] that the presence of water film on quartz surface gives higher surface conductivity. Curves 1, 2, and 3 of the same figure indicate that when moisture content is about 2 pct or higher, the quartz particles form large aggregates. These aggregates are subjected to greater centrifugal force by the revolving roll but are too heavy to be widely spread as compared with small particles.

From the above data and discussion it can be concluded that, with other factors being constant, quartz particles will be repelled farthest when their moisture content is sufficient to form thin water-films on quartz particles but not excessive to form aggregates. This condition is achieved when 48x65 mesh quartz particles contain 0.45 pct moisture, as shown in curve 4 of fig. 6.

Feed Rate: The tested data of fig. 7 indicate that, to a certain limit, galena particles are repelled farther with the decrease of feed rate. On account of the limited surface area of the grounded roll, at high feed rate some of the particles may not have the opportunity of making direct contact with the grounded roll. Furthermore, at high feed rates the charged particles must pass through a thick layer of uncharged particles resulting in interference and nullification. Thus, with the reduction of feed rate, these detrimental effects are partially eliminated.

Potential and Polarity: In regard to the effect of the magnitude of applied voltage on the behavior of mineral particles, fig. 8 shows that to a certain limit calcite particles are repelled farther with the increase of applied voltage. When applied voltage is increased the field intensity[17] is also increased and consequently the electric induction of mineral particles is facilitated. A mathematical deduction of the force on particles in terms of field intensity has been given by Hatfield[18] and advanced by Fraas and Ralston.[19]

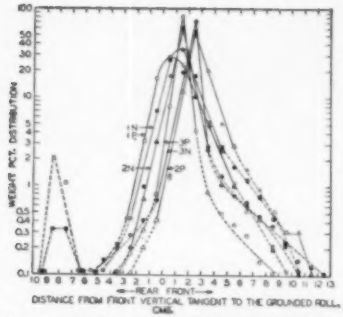

Fig. 9—Effect of the polarity of applied voltage on the behavior of garnet, fluorite, and quartz particles.

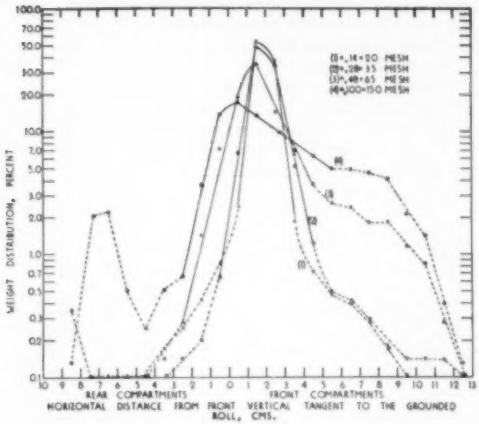

Fig. 10—Effect of particle size on the behavior of halite particles.

With respect to the effect of the polarity of applied voltage on the behavior of mineral particles, fig. 9 shows that garnet and fluorite, which were previously reported as nonreversible, are found to be respectively reversible negative and reversible positive. The contradictory results may be partially caused by the fact that mineral samples used in this paper differ from that used in the previous investigation. The other possibility may be that the distribution analyzer is more accurate to detect reversibility than the method used before.

In fig. 9, the three groups of curves *1P, 1N; 2P, 2N;* and *3P, 3N;* are, respectively positive and negative values for 100x150 mesh garnet, 48x65 mesh fluorite, and 28x35 mesh quartz particles tested with grounded roll charged at 15.5 kv.

Particle Size: From the tested data of fig. 10 it can be seen that, to a certain limit, the dispersion or spread of the weight per cent distribution curve of halite is increased with the decrease of particle size. This is due to the fact that fine particles with larger specific surface area can attain higher specific surface density of electrical charge than coarse particles. Furthermore, fine particles are less subjected to gravitational force than coarse particles, because of their difference in weight.

Summary: 1. A new apparatus termed "distribution analyzer" is used to determine the optimum condition of electrostatic separation and the electrostatic behavior of minerals under various conditions.

2. It is found that with similar surface condition, test environment and a constant feed rate of not higher than one layer of particles fed on the grounded roll, the behavior of each component mineral in a synthetic mixture or ore is roughly similar to that of the same mineral when tested alone.

3. The main factors which influence the behavior of mineral particles in electrostatic separation are: surface conductivity, of mineral particles, chemical treatment, moisture content, feed rate, potential and polarity of applied voltage, and particle size.

4. With other factors being kept constant, quartz particles are repelled farthest when their moisture content is sufficient to form thin water-film on quartz surfaces and not excessive to form aggregates.

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Organizing and Financing Cooperative Research

by Elmer R. Kaiser

Cooperative research is an established and important activity of modern business by which whole industries can advance on a broad front. The economy of jointly sponsored research as well as the limited research talent available direct attention to the advisability of projects in which a number of companies participate.

After outlining the growth of industrial research, the paper discusses one successful method for organizing and financing group research programs. A systematic procedure is advocated by which engineers and scientists can bring to management practical plans for cooperative research and development programs.

Industry Adopts Research: Seventy-five years ago Thomas A. Edison began in a modest way and with limited funds to gather about him men of various talents to form the first industrial research laboratory. His laboratory at Menlo Park, N. J., became famous for many discoveries and inventions, but possibly the most important of all may yet be the industrial research laboratory itself.

ELMER R. KAISER, Member AIME, is Assistant Director of Research, Bituminous Coal Research, Inc., Columbus, Ohio.

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Today, private industry operates 2500 research laboratories in which 135,000 persons[1] are directly and indirectly creating new processes and products and improving old ones, as well as developing new information to help keep the fountain of knowledge from running dry. In this change-making function, the scientific personnel comprises 41 pct; the technical personnel 26 pct; and the balance, 33 pct, nontechnical.

The term "industrial research" has become so broad as to include almost everything beginning with the search for and discovery of the laws governing the properties of matter and energy to the development and testing of processes, methods, and equipment. These classes of investigation commonly merge so that no sharp boundary can be traced between them. The National Research Council therefore defines industrial research as "the endeavor to learn how to apply scientific facts to the service of mankind."[1]

Currently, expenditures for research by industry are variously estimated at between $450,000,000 to $700,000,000 a year,[2] or about twice the expenditure for 1940. The President's Scientific Research Board[3] estimated that the national research and development for 1947 (excluding atomic energy) was as follows:

Industry		$ 450,000,000
Government		625,000,000
War and Navy Depts.	$500,000,000	
Other departments	125,000,000	
University		45,000,000
Other		40,000,000
	Total	$1,160,000,000

A few companies spend as much as 5 pct of their gross receipts for research, but 1½ to 2 pct is considered a good proportion by many.[4]

Shortages: Commendable as is the record of growth and accomplishment of science in industry, it is only a beginning. America and the rest of the world have merely sampled the fruits of industrial research; most industries have yet to make research the key to their future.

A top research consultant, T. A. Boyd, of General Motors, states ". . . perhaps 90 per cent of all companies still have no research laboratories."[5] For example, of more than 17,000 manufacturing concerns whose gross sales exceed $500,000 a year, less than 2500 have research laboratories. A still smaller proportion of the 184,230 manufacturing establishments in 161 industries[6] have or could afford their own research facilities. Yet, the small and medium-sized companies with less than 250 employees each, produce one half of the dollar volume of manufactured products.[7]

How are these companies to participate in and share the benefits of research?

Boyd gives two reasons for the inadequate number of laboratories in industry:

Many people still do not appreciate the importance of research, nor understand its tremendous potentiali-

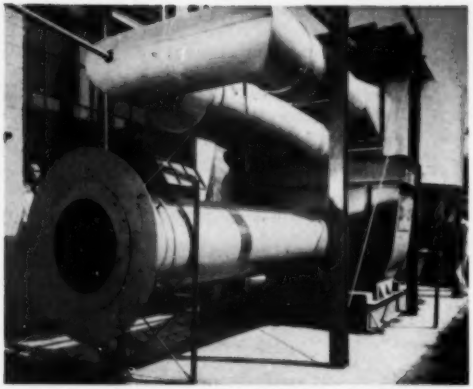

Fig. 1—Bituminous Coal Research, Incorporated, is one of the largest cooperative sponsors of research at Battelle Memorial Institute.

(This apparatus is an experimental combustor used in a project aimed at the development of a coal-driven gas-turbine locomotive. The work is conducted under the auspices of BCR's Locomotive Development Committee and financed by railroad and coal interests.)

ties. Another important reason lies in the belief that research is too costly and too complicated for any but large organizations to undertake.

Still another factor, equally vital in any attempt to increase the total research facilities rapidly, is the shortage of technically trained and experienced men and women to organize and conduct research. The demand for scientists and engineers still exceeds the supply, and will continue to do so for some time, despite the increasing numbers being recruited from the graduating classes of the colleges and universities.

Thus the combined limitations of management, money, and man power are forcing industry to look about for additional facilities, talent, and suggestions for growing research requirements.

Cooperative Research an Answer: Individual companies have a choice today whether to (*a*) found their own laboratories, (*b*) sponsor private projects at established research institutes or at independent research and development laboratories, (*c*) sponsor or conduct research cooperatively with competitors or others, and (*d*) combinations of (*a*), (*b*), and (*c*). Of course, there is always the option of not participating in research at all, which is fast becoming recognized as another form of gambling in business with the odds against success.

The first two options have been discussed in most of the many books and articles on industrial research. Less known are the potentialities of cooperatively sponsored programs and how they are organized and financed. Once such programs are established, the technical work itself proceeds much as in company laboratories.

Under joint sponsorship, research on such subjects as "The Behavior of Steels in High-Temperature Steam Piping Service," can be undertaken by a single team of researchers for a group of companies at only a fraction of the cost and man power per company that would be required if each company made the study privately. Results obtained by a group are more widely and more readily accepted, and more experience can be applied through the committee of sponsors which administers the program. This is an example of the type of research that individual companies would be glad to help support, but cannot justify doing alone for the benefit of an industry.

It is the purpose of this paper to discuss some of the factors and experience in cooperative research, including the organization of such undertakings, possible combinations of sponsorships, and dividing the cost among a sizable number of sponsoring companies. Many opportunities exist for the organization of sponsoring groups by those who will give time and effort to the work. It is a vital activity for our industrial progress and national strength in which technically trained men must take a leading role.

Research by Associations: Companies with common business problems are often members of trade associations, of which there are 11,000 in the United States. Of this number, 2600 are national and interstate organizations and 1900 are associations of business competitors in one industry.[5] "Industrial research as to production or products" is one of the principal activities of 629 associations, while 752 associations include "commercial research as to markets or marketing."

Trade associations form a logical medium for organizing and sponsoring research programs for the industries they represent. Businessmen discuss openly in meetings of such cooperative organizations many industrial matters which would be considered trade secrets in some countries. Here research is a business matter and an important factor in the ability of an industry to meet competition.

Outstanding examples of 30 associations in industry which have their own laboratories are American Gas Association, Anthracite Institute, National Lumber Manufacturers Association, Portland Cement Association, and Underwriters Laboratories.[5] Such laboratories are supported by dues from members and from additional contributions of funds and payments for special services.

Over 35 associations, including a few which operate their own laboratories, sponsor projects at universities and institutes.[6] Among the industrial laboratories established to conduct research for industry on a not-for-profit basis are Armour Research Foundation, Battelle Memorial Institute, Mellon Institute, Midwest Research Institute, and Southern Research Institute. Several hundred independent research and

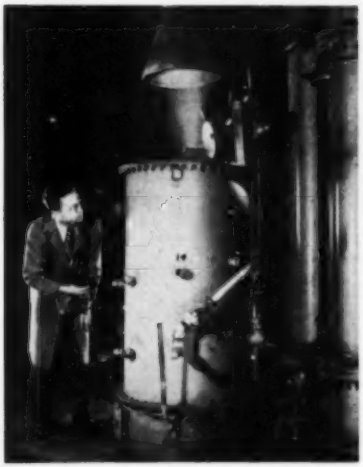

Fig. 2—As part of its program of research aimed at improvements in manufacture of gas, the American Gas Association is sponsoring at Battelle a study of fundamentals of water-gas reaction.

(This small-scale generator for production of water gas is used in research to study reactions that occur in generating bed.)

development laboratories are also available for carrying out cooperative and private programs for large and small businesses.

Among government agencies, the National Bureau of Standards, Agricultural Department, and Bureau of Mines are three of many units[4] in which industries have working arrangements for research, sponsored by industry in government laboratories. For example, a study of acid mine drainage is being sponsored at the Bureau of Mines in behalf of the coal producers by Bituminous Coal Research, Inc. The basic studies of coal carbonization of American coals is another example of joint research with support over a period of years from the American Gas Association and now continued by the Bureau.

Many professional technical societies are also active in research. The American Society of Mechanical Engineers, for example, serves as a forum and coordinating agency. Seventeen special research committees are active, and support to date of $586,000, largely from industry, reflects the confidence industry has in such work.

The American Institute of Mining and Metallurgical Engineers sponsors research through The Engineering Foundation, but industry contributes directly to projects requiring additional funds. The Committee on Research is made up largely of representatives from industry.

The medium of an existing organization is not necessary for organizing a jointly sponsored project. Two or more individual companies with a common technical problem but no facilities to investigate it, today can obtain advice from competent sources and assistance in planning a research program which they may find economical to sponsor. Producers and consumers, government and industry, universities and technical societies, trade associations and committees, manufacturers and suppliers, farmers and processors, can organize to undertake joint investigations and development work. The number of cases is increasing as scientists, engineers, and businessmen become aware of the opportunities.

Typical Topics for Cooperative Research: When the National Research Council investigated what management in nearly 100 companies expects of research,[5] the five most common replies listed in their order of rating were as follows: (1) new products, (2) maintenance of competitive technical position, (3) cutting production costs, (4) sales volume and net profit on new processes and products, and (5) serve production through development of new and improved processes.

The popularity of these expectations is confirmed by a survey on the "Research Requirements of American Industry" by the Evans Research and Development Corporation.[5] The survey revealed that 47.6 pct of industry invests funds in research to improve present products and processes, 42.3 pct to develop new products and processes in their own fields, and 14.7 pct to develop products and processes in other fields.

The types of projects appropriate for groups to sponsor are not necessarily the same as for individual companies. In some cases they depend upon the degree of initiative in the competitive race that members wish to retain. The following are generally suitable:[6] (*a*) studies of raw materials, (*b*) extension of the commercial uses of products, (*c*) standardization of analytical procedures and improvement in methods and instruments of measurement, (*d*) improvement in safety and sanitation, and (*e*) expansion of fundamental science in industry's field.

When members agree, the list can be extended with the following: (*f*) development of new products, and (*g*) development of new processes and improvement of old ones.

Manufacturers usually agree to research which raises the general quality level of an industry's products.

It is not advisable to undertake or conduct projects that will: (*a*) interfere with the competitive activities of members (*b*) have limited applicability, or (*c*) be too far ahead of industry's thinking.

The ideal situation is (*a*) the knowledge it is planned to provide is needed, (*b*) that need is widely felt, and (*c*) the results obtained will be accepted immediately and used.

Organizing a Program: *Phase 1. Concept of a Project or Program:* The background necessary to evaluate opportunities in research may be obtained through years of experience in an industry or through a formal survey. Usually an acute or chronic technical problem, with its adverse economic effects, is apparent in an industry, the solution of which at reasonable cost would spell profits. An investigation usually will disclose several such problems or needs,

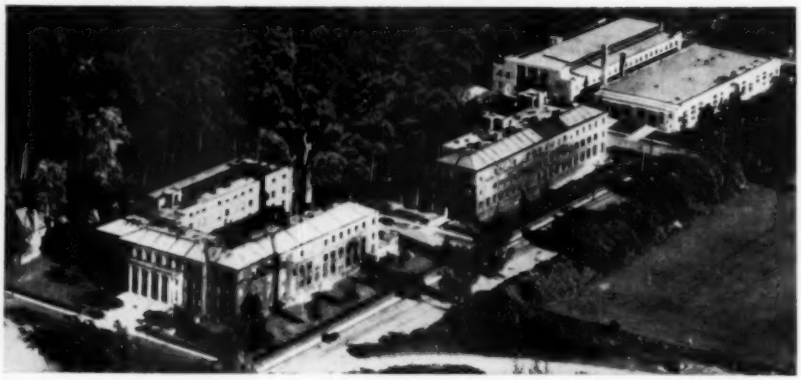

Fig. 3—An aerial view of the main buildings of Battelle Memorial Institute, Columbus, Ohio.

(Approximately 30 pct of the Institute's industrial research is for cooperative sponsorships.)

some of which are important enough to warrant a program. For example, the coal industry has urgent need for a machine that will mine coal continuously at low cost, an industrial coal burner that will automatically stoke itself and place the ashes into containers, a less expensive method of producing synthesis gas directly from coal, and a coal-burning locomotive of higher efficiency and greater service availability.

In metallurgy there are always problems in producing sound castings, in reducing corrosion, and in producing tougher metals and alloys for service at elevated temperatures. Every progressive industry has technical problems and opportunities; only dead industries do not.

Engineers and businessmen know the problems in their own industry, but may be too close to them by daily contact to see them in true perspective. An investigator from outside an industry, qualified by experience or association with research programs and business, can often see opportunities not apparent to those closely associated with daily operating matters. A fresh point of view and a knowledge of what other industries have done in similar circumstances are assets in conceiving new programs, because processes in one industry sometimes can be applied in another. Consulting research specialists are also available to suggest lines of investigation which are likely to yield valuable results. Many such specialists have laboratories and technical staffs ready to conduct all or parts of the investigations being considered.

In this initial phase, it is not necessary to make a thorough study of each opportunity or problem or how the research could be conducted, or by whom. It is better to become more thoroughly familiar with the industry under consideration, its economic justification, the companies in it, their products, volume of sales, competitive position, and the evidence of cooperation between the companies and the key individuals. Some companies are notably sympathetic toward joint efforts to improve the industry, while others prefer independent action.

Out of this preliminary study should come definite and encouraging indications of worth-while opportunities and necessary research. Clear ideas of the possibilities of a plan of research must be formulated and expressed in a written statement of the factors just discussed. The statement may constitute the report of an investigator hired for the purpose, or it may be retained by the originator of the plan for his own information and guidance. The statement may raise questions and suggest more thorough surveys. It would be premature at this stage to present a cut-and-dried outline of specific projects to be undertaken.

Phase 2. Preliminary Discussions: The product of Phase 1 is hardly more than a basis for discussion. A period of incubation is necessary to develop a plan which can be presented to industry and warrant an investment. Usually, this can best be done by the originator in personal interviews with engineers and businessmen.

Entree must usually be arranged beforehand, and it is of considerable value to be representing an organization known for good works, as a member of a committee of a technical society or trade association, employee of a research institute, independent laboratory, university, or member company in an industry.

With a knowledge of an industry's needs and problems, the originator now sets out to interview the top executives and engineers who are most likely to lend a sympathetic ear and give sincere comments and suggestions. When businessmen learn that an engineer has come to seek their advice and suggestions on some ideas he has gathered for improving (or increasing the market for) the industry's products, for reducing costs, or for developing new processes, he can be sure of a welcome.

The trade and professional associations in the industry are vitally interested in any consideration given to the industry's welfare. The directors and officers are usually glad to assist in furnishing information and in suggesting avenues of approach to key individuals. Editors of the leading journals in the industry are also well informed and interested in the industry's future.

Valuable experience and information not known outside the company or industry, all bearing directly or indirectly on the particular subject under discussion, are often made available. Reminiscence sheds light on old ideas and helps to evaluate new ones. Informal discussion, in which it is wiser to listen than talk, is the best meeting of the minds. In companies which have research facilities, the director of research should be interviewed, as he will have ideas and advice to contribute.

At this stage, it is best not to let any portion of the plan "jell" or "freeze," and one must not be too definite as it leads to fixed positions and may develop early opposition. New ideas, and that includes good ones, are usually criticized and said to be unsound and impractical until they can be organized and expressed properly. The important criterion is not whether someone believes a program can be organized or will not be supported, but why? Constructive comments and advice are obtained from only a few, but they are all important. However, it is best to listen to everyone.

In this second phase, it is usually better to have individual interviews than joint meetings as they provide more freedom of thought and expression and permit confidential information to be given to the interviewer. During the preliminary discussions, it is well to determine who is who in the industry, which executives are held in highest regard and are well known for high integrity. Interest should be developed in attending an organization meeting should the preliminary discussions disclose a sincere desire on the part of a number to proceed with planning. The auspices under which the meeting could be called must be well established so as to prevent any feeling of factionalism. It is advisable to ask who would be interested in attending an organization meeting, and which men in each company would represent (*a*) management and (*b*) engineering or research.

Many will readily see the advantage of cooperative enterprise to develop the ideas, as few companies have the financial resources available to develop a project that will benefit an entire industry as much as the developer company.

Enough executives should be interviewed to obtain comments from at least the representative units of the industry or until the pattern of comments repeats itself often enough to show definite interest in proceeding to the next step.

Preliminary discussions may be held in small technical committee meetings, which can be a great saving of time, if those present are all interested in

formulating a plan and have the necessary background and freedom of expression. This is suggested only where the originator is well known and has the full confidence of those invited. This procedure is usually followed in technical societies, except that only one project is usually considered at a time rather than a broad program.

Phase 3. Organization Meetings and Committees: When and if enough interest has been shown in the several major ideas that have been discussed, a meeting of a selected group of men can be called under the proper auspices to develop the ideas further and to take some coordinated action. Invitations should be extended to all companies, groups, and individuals who are believed to be of a cooperative frame of mind and likely to be helpful in organizing a program. A central location should be selected for the meeting and notice given at least two weeks in advance. A list of names of those invited, an agenda, and any memoranda that might well be considered beforehand should be sent with the invitations so as to keep everyone as fully informed as possible.

In a trade association the president is a logical one to call the meeting and to serve as chairman, but where no such association exists, one of the leading engineers or executives should be selected. A secretary should be designated, and minutes recorded for preparation and distribution later. The minutes are to be sent to those who attended and to others who were also invited. As the meetings are to be working meetings, it is best to limit the attendance to 25 or 30, and usually to a smaller number. However, there is danger in having too small a group and excluding some whose interest will be needed later.

The first group meetings to consider a joint research program may be either on the business or technical level. If a business meeting is called first, action may be taken on the advisability of a program, the probable cost, who should be invited to joint sponsorship, and who is to have charge of arranging the plan and program. The technical details are left to a technical advisory committee and an executive committee.

If a technical meeting is held first, objectives of the program can be outlined, a number of subjects listed, together with an estimate of cost and a suggestion for personnel organization.

In one type of program the first meeting is usually best made a technical one with the sole objective of analyzing the problems that have been suggested, and agreeing if possible on a limited selection of problems. To give point to the meeting, a talk by the originator or investigator should be presented in which a digest of the problems and a general summary of the comments is given without identity as to source.

Each of those present should be free to put forth his best comments and suggestions without committing his company. Little or no regard should be given to who would sponsor the work, except that it would be assumed it would be a joint project sponsored by the members of the industry and all will be invited to join. Little or no attention need be given in the beginning to where the work may be done and what the cost may be. It is likely if attention is given to these details, conclusions may be reached which are entirely premature.

The presence of one or two top-management executives or prominent individuals usually has a good effect in giving moral support to the meeting and encouragement to the technical men.

When more problems or projects are suggested than could possibly be undertaken, a selection must be made in a democratic manner. The topics may be listed in the order suggested during the meeting and lettered consecutively A, B, C, etc. Each of those in attendance then lists the topics in his own order of importance or preference, after which the ballots are collected and the results tabulated. In this manner twenty projects can be reduced to five or ten.

Too much caution cannot be exercised in maintaining democratic procedures and every evidence of fair consideration throughout the organization of a cooperative program. In highly competitive industries, particularly, bad blood is easy to generate as many individuals whose cooperation is highly desirable are sensitive to their prerogatives.

Individuals and companies may have dissimilar ideas and interests about what should be done, and how, but agreement can be reached directly on some matters, and a suitable compromise usually can be reached on others. Fortunately, there generally are so many noncontroversial problems to solve in an initial program that others can be laid aside.

The culmination of Phase 3 is a written statement of the projects proposed, together with a list of those who participated in the meetings or those who endorse the program. A short statement of each project, a justification for research on it, and an estimate of the annual expenditure recommended for it will be required. The estimate of annual cost can be made by research directors or consultants with reasonable accuracy. Predictions of the time that will be required to complete a research project are of limited value and usually unnecessary. If the work is progressing satisfactorily, sponsors often find it desirable to continue on at least a year-to-year basis.

The statement should be written for management as a clear recommendation of the group of engineers or executives who have participated in the discussions and meetings up to this time.

Phase 4. Financing the Program: Those who will have the most direct interest in a research program are the logical sponsors and administrators of it. The public is the ultimate beneficiary, of course, but the public expects to pay for research indirectly through the purchase of goods and services. Notable exceptions are the public contributions to research on cancer and infantile paralysis.

Many combinations of interest have financed research in the past. The sponsoring companies need not all be in the same industry. Two or more units in each of the following classifications or in two or more of these groups comprise most of the cases:

1	Raw material producers	8	Government departments
2	Manufacturers	9	Technical societies
3	Transporters	10	Research associations
4	Trade associations	11	Research institutions
5	Granges	12	Universities and colleges
6	Consumers and users	13	Research foundations
7	Private individuals	14	Endowments

Numerous examples might be cited. For instance, Groups 1 and 3, comprising coal companies and railroads, are joint sponsors of a coal-mining-machine development under the Mining Development Committee of Bituminous Coal Research, Inc. Groups 2 and 10, comprising spreader-stoker manufacturers and the research agency of the bituminous-coal industry, are sponsoring a study of combustion in cooperation with one or two stoker users (Group 6),

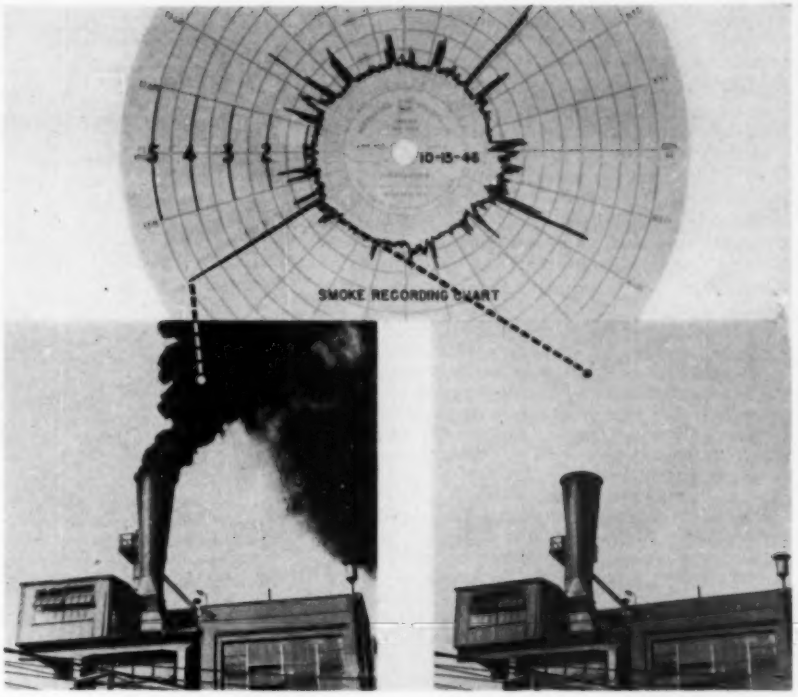

Fig. 4—Graphic proof of the effectiveness of modern overfire jets in eliminating smoke from boiler plants.

(Left illustration shows smoke emission without jets in operation; at right, 45 sec after jets were turned on. The design and application of these modern overfire jets were effected under the sponsorship of Bituminous Coal Research, Inc., the national research agency of the bituminous-coal industry.)

and a research institution, (Group 11). Groups 2, 6, 9, and 11, consisting of manufacturers, users and the ASME, are sponsoring research on the metallurgy of high-temperature steam piping at an established contracting laboratory.

A decision on the selection of participants sets the stage for a membership plan, on the basis of which the solicitations are to be made. The simplest plan is for each sponsor to pay the research laboratory the sum agreed upon and to have the laboratory pay all expenses. No contracts between sponsors, nor a special corporation, are thus required. The payments and any incidental expenses of their representatives are charged off as current operating expenses by the sponsors.

Complexities, such as changing membership, necessity for paid employees, sponsorship at several laboratories, and commercialization of inventions, necessitates the formation of a more formal organization. A corporation not-for-profit is the usual type as there is rarely any intention to declare dividends on income that may be derived. The conduct of the program requires additional dues annually from members, and any income from royalties is used to supplement the dues.

Membership dues, income from various sources, investments, and in general, property of corporations not-for-profit, are free from taxation. The state and federal regulations governing such corporations should be consulted carefully as there are other restrictions besides nonpayment of dividends.

Of course, a corporation for profit can be organized and stock issued for funds invested. The general preference in industry is the nonprofit corporation because the contributions can be written off as operating expenses rather than continued on in the books as capital expenditures. Any indirect profits are to be made by the sponsor companies themselves, and they expect to use the research results without paying royalties.

In drawing up a charter and by-laws, copies of those in effect by similar corporations can be consulted and adapted by a lawyer. A typical object and purpose is:

To encourage, foster, and promote in any lawful manner the utilization of (products of industry) and of related equipment for and method of production, preparation, or utilization of (product of industry) by means of study, research, education, and other effort, or by developing equipment for and methods of production, preparation, or utilization of (product of industry) or any other lawful purpose.

All contributing companies may be classed as members, with voting power in proportion to total contributions to date. Powers and duties are vested in a board of directors, half of which are elected each year by the members. Officers elected each year by the board include a president, vice-presidents, secretary, and treasurer. In a small organization, the directors and officers are selected from the membership and serve without pay.

The research program may be outlined and administered by a committee which reports to the president and board. When the size of the program

warrants, a director of research and an administrative staff may be employed. As a first approximation, an administrative cost of 10 pct is reasonable.

Basis of Participation: How much money shall each member company contribute? Shall each pay equally or in proportion to size of company?

In theory, an equitable proportioning of the cost is determined by the opportunity each participant has to benefit. Improvements in a product would benefit a large manufacturer financially more than a smaller one because he manufactures and sells more of them. Payments prorated on the basis of sales volume are common. In this way companies ranging in size by as much as 100 to 1 can participate on a basis that is fair to all. In developing a new product, each participant would have a more nearly equal opportunity, in which case equal assessments might be made. This is also best when the companies are on a nearly equal footing in an industry, or where the total amount from each company is small. A compromise between proportionate and equal payments, in which a minimum and maximum fee are set, may satisfy the parties.

Quarterly or annual payments are convenient and are commonly made in advance.

Membership pledges or agreements can be simple signed pledges to contribute X-dollars a year in advance for Y-years for reports of research on subjects jointly agreed upon. More explicit legal documents can be executed if desired, binding the members and the research organization, but business firms prefer annual contributions without commitments to pay beyond one year at a time.

The Time Factor: Months and years of missionary work and the investment of thousands of dollars are required to organize the first research program in a sizable industry. Once the organization has been set up, and its value demonstrated, membership can be maintained and extended more easily. No rules can be given but a few examples will illustrate:

Case 1: A program administered by a Mining Development Committee of the bituminous-coal industry's research agency, and sponsored by 71 coal companies, railroads, and land companies. Objective: The designing of a machine for cutting and loading coal continuously in underground mining.

Phase 1	Date of conception	June, 1946
Phase 2	Preliminary discussions	7 months
Phase 3	Organization meetings	3 months
Phase 4	Financing	11½ months
	Initial fund raised	$258,000

Case 2: A program administered by a Spreader Stoker Research Committee and sponsored by seven stoker manufacturers and the bituminous-coal industry's research agency. Objective: To improve the utilization of coal on spreader-type stokers.

Phase 1	Date of conception	October, 1947
Phase 2	Preliminary discussions	4 months
Phase 3	Organization meetings	2 months[a]
Phase 4	Financing	4 months[a]
	Initial fund raised	$14,000

[a] Partial overlapping.

Case 3: A program administered by a Locomotive Development Committee of the bituminous-coal industry's research agency, and sponsored by nine railroads and five coal companies. Objective: To develop a coal-burning gas-turbine railroad locomotive.

Phase 1	Date of conception	Summer of 1944
Phase 2	Preliminary discussions	6 months
Phase 3	Organization meetings	4 months
Phase 4	Financing	1 month
	Initial fund raised	$1,113,000
	Total funds pledged	$2,754,000

Case 4: A program administered by a joint committee and sponsored by 14 stove manufacturers and the bituminous coal industry's research agency. Objective: To develop principles for smokeless stoves, ranges, and water heaters.

Phase 1	Date of conception	June, 1940
Phase 2	Preliminary discussions	5 months
Phase 3	Organization meetings	4 months
Phase 4	Financing	4 months
	Initial fund raised	$70,600
	Funds to date	$180,700

Conclusion: Organizing and financing cooperatively sponsored research programs is an activity of management that engineers are well equipped to conduct. By increasing the number of participants in a program, the benefits of research are extended, and the research talent conserved. The procedures are systematic and do not include high-pressure sales tactics.

A few words of caution may be advisable. Short cuts in the steps outlined, such as expecting to organize a program in one meeting, are often doomed to failure. Personal visits and trips are far more effective than mimeographed letters and outlines, and are necessary for results.

Patience and diligence, optimism, and confidence are vital because the slowness in which decisions are received may be discouraging. The author has never found organization work or financing to proceed for long on the momentum of a few favorable decisions of prospective sponsors. Hard work is necessary at every step, but the rewards of accomplishment are indeed satisfying to the organizer and to the participants.

In the words of Prof. Warren K. Lewis:[10]

I feel that there is an unsolved problem as to how to handle the research needs of a small business, or at least one phase of those research needs, and that is the problem of competitive research in contradistinction to co-operative research. Co-operative research has been extraordinarily successful and has a large future ahead of it. By and large, there aren't many obvious rocks in the road.

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Operating Data for a Bird Centrifuge

by A. C. Richardson and Orville R. Lyons

Operating data is presented for a Bird centrifuge used to dewater coal treated at one preparation plant. The data include: (1) percentages of solids in centrifuge feed, cake, and effluent and the plant circulating water for a period of four consecutive operating shifts, and (2) screen-sizing data for a number of centrifuge feed, cake, effluent, and combined cake-effluent samples. An evaluation is included of the relationships between fine solids in the centrifuge feed and the total moisture content of the centrifuge cake.

THE Coal Division of Battelle Memorial Institute, during the course of an investigation conducted for a coal producer, carried out extensive sampling of the fine-coal section of a preparation plant. The information obtained during the testing period was used to evaluate the overall plant operation and provided the basis for recommended changes in the plant flowsheet. Included in the information were data for the results obtained when using a Bird centrifuge to dewater the particular coal being treated. At the completion of the investigation, the coal producer gave Battelle permission to publish the operating data for the Bird centrifuge in order to make this information available to the coal industry. The authors wish to take this opportunity to express their appreciation to this company and its officials for their willing cooperation in the matter of publication.

A. C. RICHARDSON, Member AIME, is Supervisor, Minerals Processing Division, Battelle Memorial Institute, Columbus, Ohio, and ORVILLE R. LYONS, Member AIME, is Coal Preparation Engineer, Heyl and Patterson, Inc., Pittsburgh, Pa.

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TP 2818 F. Discussion (2 copies) may be sent to Transactions *AIME before April 30, 1950. Manuscript received Oct. 14, 1949.*

Description of Preparation Plant: The preparation plant on which the data were obtained is primarily a jig plant. The coal being washed is a low inherent moisture, friable, Pocahontus-type. The washed coal from the jig is dewatered and sized at ¼ in. by a Parrish-type dewatering and sizing screen. The minus ¼-in. coal and the underflow water from the shaker screen flow to a sludge tank where, after the coal settles, the coal is conveyed by a flight conveyor into a pump sump. The pump, a 300-gpm sand pump, delivers a mixture of fine coal and water to a Bird centrifuge which produces a dewatered cake and an effluent. At the time when the investigation was made, the centrifuge effluent was returned to the sludge tank and the cake product was mixed with a coarser size of coal prior to loading into railroad cars. The semiclarified water from the sludge tank overflowed into a pump sump from which it was pumped to the jig.

As originally designed, the preparation plant was intended to operate with a closed water system but it was found, soon after operations started, that for various reasons this could not be done. In order to provide for removal of excess water from the system and to aid in maintaining the desired minimum percentage of solids in the circulating water, the plant was operated to provide from the sludge tank a continuous overflow of water and fine solids that was sent to waste.

The centrifuge feed represents approximately 35 tons per hour of minus ¼-in. coal in the form of a pulp averaging 38 pct solids.

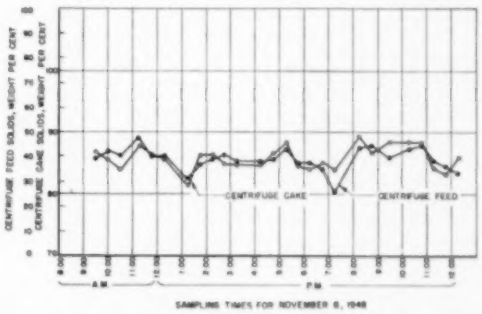

Fig. 1—Relation between solid contents of Bird centrifuge feed and cake.

Sampling Procedure: In order to obtain basic data for determining the relationships existing in the fine-coal section of the preparation plant, samples were obtained for: (1) the centrifuge feed, (2) the centrifuge cake, (3) the centrifuge effluent, and (4) the circulating water. The sampling was arranged to start on a Monday, in order to begin operations with a sludge tank full of fresh water, and was continued long enough to allow sufficient time for fine solids to build up within the water system or for a two-day, four-shift operating period. All samples were taken at half-hour intervals except when plant stoppages occurred. After a stoppage the plant was

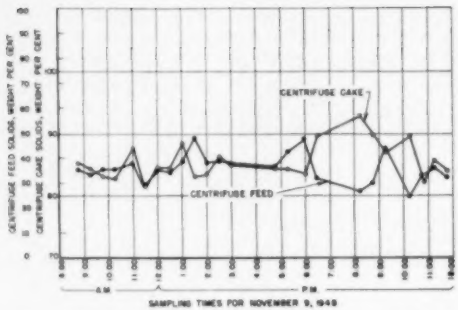

Fig. 2—Relation between solid contents of Bird centrifuge feed and cake.

allowed to operate for 15 min prior to the resumption of sampling.

The pulp samples, or the centrifuge feed, centrifuge effluent, and the circulating water, were obtained by momentarily diverting the full, pulp stream. The centrifuge-cake samples were obtained by cutting out, by means of a shovel, small increments of the cake from the total cake as it was being conveyed away from the centrifuge. These increments were taken over a period of 2 to 3 min and a sufficient number of increments were taken to provide a total individual sample weight of approximately 35 lb. The pulp samples were placed in glass jars and sealed. The centrifuge-cake samples were placed in 10-gal garbage cans which were weighed prior to and immediately after being filled with the sample.

The glass jars and garbage cans containing samples were labeled inside and out, were securely fastened to prevent spillage, and were shipped to Battelle.

Centrifuge Performance: Table I contains the data for percentages of solids and total moisture for all of the samples obtained. A study of table I reveals that: (1) the centrifuge-feed solids varied from 25.3 to 47.9 pct by weight with a normal value of approximately 38 pct, (2) the centrifuge-cake solids varied from 81.5 to 92.8 pct by weight with a normal value of approximately 85 pct, (3) the centrifuge-cake total moisture varied from 7.2 to 18.5 pct with a normal value of approximately 15 pct, (4) the centrifuge-effluent solids varied from 1.5 to 5.5 pct by weight with a gradual increase in the percentage of solids throughout most of the sampling period, and (5) the circulating-water solids varied from 3.9 to 11.6 pct by weight with a gradual increase in the percentage of solids throughout most of the sampling period.

Figs. 1 and 2 show the relationship existing between the solid contents of the centrifuge feed and the centrifuge cake. A study of these figures indicates that the solid content of the centrifuge cake varied directly with the solid content of the centrifuge feed until approximately 5:00 p.m. on November 9, at which time the relationship changed and the solid content of the cake started to vary inversely with the solid content of the feed. The observations contained in table I show that at 5:00 p.m. the pump used to elevate fine coal from the sludge tank to the Bird centrifuge started to give trouble and that pumping operations were erratic from 5:00 p.m. until midnight. Fresh water was added to the sludge tank, in considerable quantity, almost continuously from 5:00 p.m. until midnight, supposedly to help the fine-coal solids flow into the sump, from

Table I. Plant-performance Data

Date	Sampling Time	Bird Centrifuge: Feed Solids, Wt, Pct	Bird Centrifuge: Cake Solids, Wt, Pct	Bird Centrifuge: Cake Moisture,* Wt, Pct	Bird Centrifuge: Effluent Solids, Wt, Pct	Circulating-water Solids, Wt, Pct	Observations
Nov. 8	9:30 a.m.	39.8	86.9	13.1	1.9	3.9	Preparation plant started at 8:00 a.m.
	10:00 a.m.	42.3	85.6	14.4	2.1	4.5	
	10:30 a.m.	40.8	84.5	15.5	2.6	5.2	
	11:15 a.m.	48.1	88.1	11.9	2.2	6.1	
	11:45 a.m.	40.3	86.6	13.4	1.7	6.5	
	12:15 p.m.	40.4	85.9	14.1	1.9	6.7	
	1:15 p.m.	30.6	81.6	18.4	3.0	6.5	
	1:45 p.m.	37.3	86.3	13.7	3.4	6.2	
	2:15 p.m.	39.2	86.3	13.7	3.1	6.2	
	2:45 p.m.	40.6	85.2	14.8	3.2	6.3	
	3:15 p.m.	37.9	84.9	15.1	3.1	6.5	
	4:15 p.m.	37.7	84.7	15.3	3.3	6.1	Plant shut down between shifts from 3:30 p.m. until 4:00 p.m.
	4:45 p.m.	39.1	86.7	13.3	3.2	6.7	
	5:15 p.m.	43.5	88.6	11.4	3.8	6.8	
	5:45 p.m.	37.1	84.7	15.3	3.4	7.0	
	6:15 p.m.	37.6	84.4	15.6	3.5	7.4	
	6:45 p.m.	35.2	85.0	15.0	3.8	7.8	
	7:15 p.m.	26.0	84.2	15.8	4.1	7.3	
	8:15 p.m.	43.0	89.4	10.6	3.4	7.9	Plant waiting on coal 7:30 p.m. until 8:00 p.m.
	8:45 p.m.	44.2	86.7	13.3	3.4	8.1	
	9:30 p.m.	39.4	88.5	11.5	3.3	8.5	
	10:15 p.m.	42.8	88.5	11.5	3.4	8.4	
	10.45 p.m.	44.4	88.2	11.8	3.4	9.2	
	11:15 p.m.	38.6	84.3	15.7	4.0	8.7	
	11:45 p.m.	35.9	83.7	16.3	4.1	8.4	
Nov. 9	12:15 a.m.	33.5	86.2	13.8	4.2	7.8	
	8:45 a.m.	35.0	84.8	15.2	4.6	9.2	
	9:15 a.m.	33.3	84.2	15.8	5.0	9.2	
	9:45 a.m.	34.9	83.1	16.9	5.3	9.0	
	10:15 a.m.	35.0	82.4	17.6	5.5	9.6	
	11:00 a.m.	38.1	87.5	12.5	5.0	10.3	
	11:30 a.m.	29.4	81.5	18.5	5.4	10.0	
	12:00 noon	35.3	84.0	16.0	5.4	11.3	
	12:30 p.m.	33.9	84.2	15.8	5.2	9.9	
	1:00 p.m.	38.6	88.3	11.7	5.0	10.0	
	1:30 p.m.	43.6	83.0	17.0	4.0	10.5	
	2:00 p.m.	38.2	83.3	16.7	5.5	11.2	
	2:30 p.m.	39.0	86.1	13.9	5.1	10.4	
	3:00 p.m.	38.1	85.1	14.9	5.1	10.8	Plant shut down between shifts from 3:30 p.m. until 4:30 p.m.
	4:45 p.m.	36.6	84.4	15.6	5.4	10.4	
	5:15 p.m.	42.7	84.4	15.6	4.7	10.8	
	6:00 p.m.	47.9	83.3	16.7	5.3	11.0	Fire hose connected to 300 gpm sand pump and fresh water added to insure continuous flow.
	6:30 p.m.	32.1	89.5	10.5	3.0	11.6	
	8:15 p.m.	26.9	92.8	7.2	1.4	10.4	Erratic centrifuge-feed conditions from 5:00 p.m. until 12:00 midnight.
	8:45 p.m.	30.5	89.5	10.5	1.5	9.8	
	9:15 p.m.	44.7	87.1	12.9	3.6	9.7	
	10:15 p.m.	25.3	89.8	10.2	1.7	9.7	
	10:45 p.m.	33.5	82.0	18.0	5.2	9.6	
	11:15 p.m.	36.5	85.6	14.4	3.6	10.0	
	11:45 p.m.	33.3	84.1	15.9	3.7	10.7	

* Seam moisture is approximately 1.5 pct.

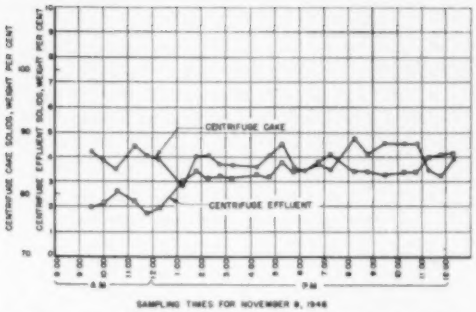

Fig. 3—Relation between solid contents of Bird centrifuge cake and effluent.

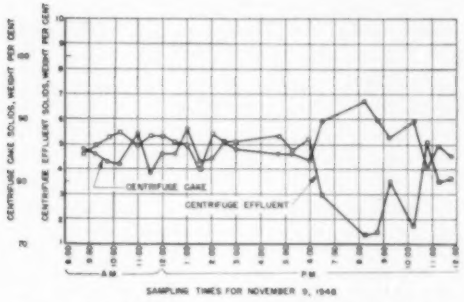

Fig. 4—Relation between solid contents of Bird centrifuge cake and effluent.

Table II. Screen-sizing Data for Bird Centrifuge Feed Samples Taken on November 9, 1948

Mesh	Weight, Per Cent					
	11:30 a.m.	12:00 Noon	1:30 p.m.	6:00 p.m.	8:15 p.m.	8:45 p.m.
—3 + 6	14.6	14.8	12.0	8.1	15.7	16.7
—6 + 10	22.6	19.5	27.9	23.8	40.3	26.3
—10 + 14	8.9	7.2	12.7	15.0	13.9	10.9
—14 + 20	10.8	9.5	13.4	14.7	6.2	11.4
—20 + 28	10.0	9.7	9.9	11.7	6.4	8.9
—28 + 35	5.7	7.2	5.9	6.7	3.3	5.7
—35 + 48	5.9	8.1	5.1	6.5	3.9	5.9
—48 + 65	4.3	6.1	3.4	3.6	2.7	4.1
—65 + 100	3.4	4.8	2.4	2.5	2.1	3.2
—100 + 150	2.3	3.0	1.3	1.5	1.4	1.8
—150 + 200	1.9	1.7	1.1	1.3	1.0	1.2
—200	9.6	8.4	4.9	4.6	3.1	3.9

which the centrifuge feed was withdrawn by means of the sand pump. Addition of fresh water to the sludge tank apparently diluted the feed pulp to some extent, as the 10 feed samples taken from 11:00 a.m. until 5:00 p.m. had an average of 37.6 pct solids, while the 10 feed samples taken from 5:00 p.m. until 11:45 p.m. had an average of 35.3 pct moisture, but this difference is not great enough to be responsible for the reversed relationship.

Table II contains screen-sizing data for samples of the centrifuge feed taken before and after 5:00 p.m. on November 9. A study of the data contained in table II reveals that the minus 200-mesh material in the centrifuge feed averaged 7.6 pct in the samples obtained prior to 5:00 p.m. and averaged only 3.9 pct in the samples obtained after 5:00 p.m. This marked reduction in the amount of minus 200-mesh solids in the centrifuge feed was evidently the result of a classifying action created by overflowing the sludge tank and wasting excess water and entrained coal.

From the evidence available, therefore, the inverse relationship between the cake solids and the feed solids after 5:00 p.m. on November 9 appears to be the result of centrifuging a feed from which

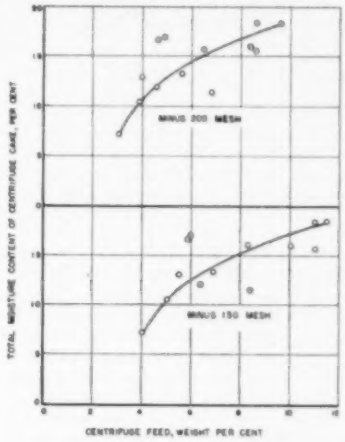

Fig. 5—Relation between fine solids in Bird centrifuge feed and moisture content of Bird centrifuge cake.

approximately 50 pct of the minus 200-mesh solids had been removed by classification. It seems indicated that the moisture content of the centrifuge cake obtained after 5:00 p.m. on November 9 was considerably lower than what it would have been if the additional minus 200-mesh material had been present in the centrifuge feed.

Figs. 3 and 4 show the relationship existing between the solid contents of the centrifuge cake and the centrifuge effluent. A study of the figures indicates, that, in general, the solid content of the centrifuge effluent varied inversely with the solid content of the centrifuge cake and that this inverse relationship was most pronounced from 5:00 p.m. until midnight on November 9. Evidently the centrifuge-cake moisture content and the effluent-solid content are proportional to the amount of fine solids in the centrifuge feed. The data presented in table III were used in the preparation of fig. 5 which shows the relationship existing between the minus

Table III. Data for Fine Solids in the Centrifuge Feed and for Moisture in Centrifuge Cake

Sample	Weight, Per Cent		Centrifuge Cake Moisture, Per Cent
	Minus 150-mesh	Minus 200-mesh	
Nov. 8, 9:30 a.m.	5.5	4.0	13.1
11:15 a.m.	6.4	4.6	11.9
1:15 p.m.	11.0	8.6	18.4
4:45 p.m.	6.9	5.6	13.3
7:15 p.m.	11.0	8.6	15.8
9:30 p.m.	8.4	6.8	11.5
Nov. 9, 11:30 a.m.	11.5	9.6	18.5
12:00 Noon	10.1	8.4	16.0
1:30 p.m.	6.0	4.9	17.0
6:00 p.m.	5.9	4.6	16.7
8:15 p.m.	4.1	3.1	7.3
8:45 p.m.	5.1	3.9	10.5
11:45 p.m.	8.3	6.5	13.9

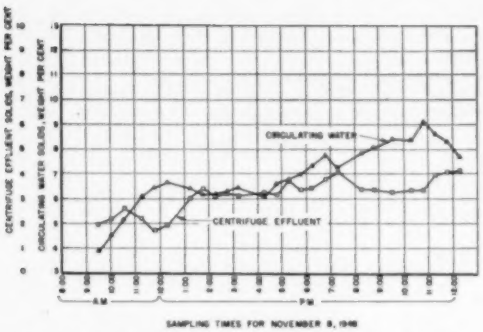

Fig. 6—Relation between solid contents of Bird centrifuge effluent and circulating water.

150-mesh and minus 200-mesh solids in the centrifuge feed, in per cent, and the total moisture content of the centrifuge cake, in per cent. The moisture content of the centrifuge cake varies with the percentage of fine solids in the centrifuge feed but the relationship is not exact enough to allow a more definite conclusion than that the cake moisture apparently can be reduced to less than 10 pct total moisture by the removal of fine solids from the feed material.

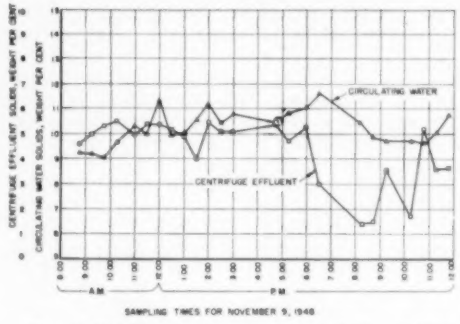

Fig. 7—Relation between solid contents of Bird centrifuge effluent and circulating water.

Table IV. Infrasizer Data for Centrifuge Effluent

Cone No.	Approximate Average Size of Solids, Microns	Effluent Solids, Wt, Pct
1		0.1
2		0.2
3		0.7
4	74	1.7
5	48	2.3
6	28	1.6
Bag	14	93.4
Total		100.0

Figs. 6 and 7 show the relationship existing between the solid contents of the circulating water and the centrifuge effluent. A study of figs. 6 and 7 indicates that, in general, the solid content of the circulating water varied with the solid content of the centrifuge effluent. The return of effluent solids to the sludge tank apparently was responsible for the buildup of solids in the circulating water.

In order to determine the size composition of the effluent solids, one of the samples was infrasized by means of a Haultain infrasizer. Table IV contains the results of the infrasizer test. It is evident that the effluent solids are extremely fine and that their removal from the circulating-water system, which appears necessary on the basis of the buildup condition determined, presents a difficult problem.

Table V. Comparative Screen-sizing Data for Bird Centrifuge Feed and Product Samples

Mesh	9:30 A.M. Nov. 8			6:00 P.M. Nov. 9		
	Feed Wt, Pct	Cake Wt, Pct	Composite Cake Plus Effluent Wt, Pct	Feed Wt, Pct	Cake Wt, Pct	Composite Cake Plus Effluent Wt, Pct
—3 + 6	21.7	24.0	23.3	8.1	11.8	10.9
—6 + 10	26.2	20.7	20.1	23.8	17.2	15.9
—10 + 14	9.0	14.3	13.9	15.0	13.9	12.9
—14 + 20	8.4	7.0	6.8	14.7	9.5	8.8
—20 + 28	7.7	6.9	6.7	11.7	8.6	7.9
—28 + 35	5.3	5.5	5.3	6.7	6.7	6.2
—35 + 48	6.2	3.7	3.6	6.5	6.4	5.9
—48 + 65	4.4	4.4	4.3	3.6	5.8	5.4
—65 + 100	3.6	3.7	3.6	2.5	5.4	5.0
—100 + 150	2.0	2.1	2.0	1.5	3.0	2.8
—150 + 200	1.5	1.9	1.9	1.3	4.6	4.2
—200	4.0	5.8	8.5	4.6	7.1	14.1

A study of this nature would be incomplete without some discussion of the amount of degradation that occurs when dewatering fine coal by means of a Bird centrifuge. Table V contains comparative screen-sizing data for samples of the centrifuge feed and cake and for composite samples of the cake plus the effluent solids. It is evident that considerable degradation does occur. The sample for 9:30 a.m. November 8 shows an increase in the minus 200-mesh fraction from 4.0 pct in the feed to 8.5 pct in the combined cake-effluent product, while the sample for 6:00 p.m. November 9 shows an increase from 4.6 pct in the feed to 14.1 pct in the combined cake-effluent product. There are some inconsistencies in the screen analyses, in that in both cases the minus 3-plus 6 mesh fraction of the combined product represents a larger percentage than that shown possible by the screen analyses of the feed. However, these inconsistencies are the result of a time lag in the flow of material through the centrifuge when taking instantaneous samples and the general conclusion that considerable degradation occurs is still valid.

Summary: This paper presents operating data for a Bird centrifuge working on the coal treated at one preparation plant. For this particular installation, when treating a minus ¼-in. feed in the form of a pulp that had a normal value of approximately 38 pct solids, the machine had the following characteristics: (1) the centrifuge-cake total moisture had a normal value of approximately 15 pct, (2) the centrifuge-effluent solids varied from 1.5 to 5.5 pct by weight with a gradual increase in the percentage of solids throughout most of the sampling period, and (3) the moisture content of the centrifuge cake varied to a considerable extent with the percentage of minus 150-mesh and minus 200-mesh solids in the centrifuge feed.

It would be helpful to the coal industry if similar data could be published for other feed conditions or for other types of equipment. It is the authors' hope that operators possessing such information will make it available to the industry in the near future

Production of Graded Glass Sand by Grinding and Classification

In a laboratory study of grinding and classification of silica sand, a satisfactory means of producing the medium-fine specification sand desired by producers of flint-glass containers was developed. The best procedure consisted of ball-mill grinding and double classification, which gave a properly sized sand containing less iron than the feed.

by M. M. Fine

THE problem of producing a uniform, medium-fine sand for glass-furnace feed has been of interest to the glass-container industry for many years. In the present investigation of the problem, conducted by the Bureau of Mines in cooperation with the Owens-Illinois Glass Co., Alton, Ill., a satisfactory method of producing such sand was developed. The investigation involved continuous-grinding studies of glass sand with flint and porcelain pebbles, steel balls and rods, and a simultaneous study of classification.

M. M. FINE, Member AIME, is Metallurgist, Rolla Branch, Metallurgical Division, Bureau of Mines, Rolla, Mo.

Laboratory work for this report was completed April 1949. This paper presents the results of work done by the Rolla, Mo., Branch, Metallurgical Division, of the Bureau of Mines.

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The most satisfactory procedure consisted of grinding in a pebble or ball mill followed by a double classification: hydraulic classification to separate the oversize for return to the grinding unit and mechanical classification to remove fines and impurities from the ground sand. The proper application of this flowsheet usually yielded a graded sand containing less iron than the feed even when grinding was done in a steel-lined ball mill.

The amount of sand ground into the so-called "dusting" sizes was, in all cases, less than 10 pct. Experiments were made which showed that this very fine sand could be beneficiated to a usable material.

Background

The advantages in utilizing a uniform, medium-fine sand as a feed for glass furnaces has motivated a considerable amount of research on the problem of producing such sand by glass producers and others. For the glass producer, the faster melting rate of such a graded sand offers the possibilities of reduced fuel costs and increased furnace capacity. In addition, there is the likelihood of improving the quality of the melt and thereby reducing the number of seeds (air bubbles) and stones (unmelted grains) in the finished ware.

Although the melting rate of a glass batch depends on other factors in addition to the grain size of the sand, there is experimental evidence to show that grain size influences the melting rate considerably. Potts, Brookover, and Burch[1] found that when the raw materials (sand, soda ash, and limestone) were always matched in size, there was a great decrease in the melting time as the grain size of the materials was decreased. They found, too, that when the grain sizes of the soda ash and limestone were held constant at 40 to 60 mesh, the minimum melting time occurred when 40 to 60 mesh sand was used. The increase in melting time with decrease in grain size beyond 40 to 60 mesh was due, they postulated, to accelerated demixing or separation of batch during melting as the grain size of the sand decreased in relation to the grain size of the other materials.

An earlier investigator[2] found that the highest rate of melting and purifying ordinary glass was obtained when medium-size sand (0.385 to 0.120 mm, correspondingly roughly to 40 to 120 mesh) was used. Although batches containing smaller-size sand were found to fuse more rapidly, the time required to remove air bubbles from such melts actually increased the total duration of the process. Coarse-grained sand, it was stated, increased greatly the times of both melting and purifying.

Although the laboratory data are difficult to confirm on a plant scale, observations by the Owens-Illinois Glass Co. indicate a higher furnace capacity of better-quality ware when a medium-fine sand is used in the batch.

During the recent war, the glass producer was inclined to overlook a considerable variation in raw materials in the interest of higher production. For instance, table I shows the wide range in the amounts of various screen fractions in sand supplied by one producer to the Terre Haute plant of Owens-Illinois. The sand, from the northern Illinois St. Peter sandstone, represented material used by that plant over a period of a few years during the war.

Since the war, the glass industry has insisted on

Table I. Variation in Amounts of Sand Screen Fractions during War

Size, Mesh	Range in Amount of Size, Per Cent
Plus 20	0.0 - 0.3
20 to 30	1.2 - 7.2
30 to 40	16.4 - 50.6
40 to 50	15.4 - 43.5
50 to 60	5.0 - 16.6
60 to 80	3.0 - 25.2
80 to 100	0.8 - 13.4
100 to 140	0.6 - 13.8
Minus 140	0.2 - 11.9

Table II. Postwar Sand Shipments from Two Producers, Screen Analyses

Size, Mesh	Range in Amount of Size, Per Cent	
	Sand *A*	Sand *B*
Plus 20	0.0 - trace	0.0 - trace
20 to 30	1.9 - 3.4	0.4 - 1.7
30 to 40	40.7 - 50.3	2.8 - 6.9
40 to 50	30.5 - 46.4	13.6 - 22.4
50 to 60	5.6 - 7.5	12.0 - 16.3
60 to 80	2.8 - 7.6	29.4 - 37.4
80 to 100	0.4 - 1.8	12.0 - 13.9
100 to 140	trace - 0.5	8.5 - 10.3
Minus 140	trace - 0.1	4.4 - 6.3

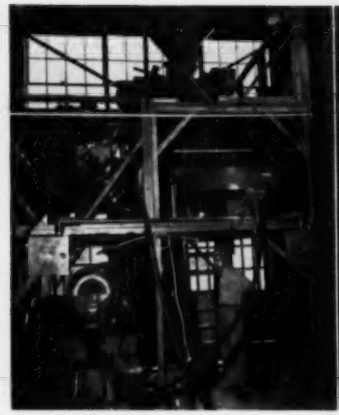

Fig. 1—Grinding and classification unit.

Fig. 2—First hydraulic classifier.

greater uniformity in glass sand, and sand *A*, table II, represents postwar shipments from another sand producer in the Illinois field. Wide ranges in the amounts of each screen fraction have been considerably reduced, but the sand is still regarded by the Owens-Illinois Co. as too coarse for an ideal glass sand.

A sand nearer the preferred screen analysis is sand *B* in table II. The sand (shipments from a deposit in eastern Missouri) corresponds to what is indicated by experience and experimental data to be desirable. Unfortunately, the operation is a small one and can supply only the sand requirements of the Alton plant.

Sand-preparation Practice

Although the American Ceramic Society and the National Bureau of Standards have established a recommended grading for glass sand, manufacturers have not adhered to the recommendation.[8] The practice in glass-sand preparation is directed, for the most part, to recovering the sand grains at the natural size in which they occur in the sandstone. Although the flowsheet may vary in different plants, the process is essentially one of liberating the individual grains in the sandstone, washing to remove excess fines and impurities and screening on about 20 mesh to separate any oversize. This is particularly true of most of the operations in the loosely cemented St. Peter sandstone of northern Illinois[5] and Missouri.[6]

In other areas, where more firmly consolidated sandstones are found, more rigorous treatment is required to liberate the individual sand grains. The operations in the Pennsylvania and West Virginia fields[9] have standardized on the chaser mill or wet-pan grinder as a fine crusher before washing. In a very few installations, a tube or ball mill has been used to supplement the action of jaw crushers and washers in disintegrating compact sandstones.[7, 8] In none of these installations, however, is the grinding unit used to prepare a specification sand for the glass industry. Whatever comminution takes place is incidental to the objective of freeing the individual grains in the sandstone.

A plant in California that was recently completed and placed in operation[9] does prepare a fine-glass sand by ball-mill grinding. This plant, the Pacific Coast Division of Owens-Illinois Glass Co., near Corona, Calif., is producing glass sand from an arkose sand in clay. The sand is scrubbed in various devices and then separated at 40 mesh in revolving screens. The primary undersize is suitable for colored glass only and is treated further in a separate circuit. The oversize is reduced to minus 40 mesh in a ball mill in closed circuit with a mechanical classifier and another revolving screen. The latter product, after washing, drying, and magnetic separation, is used in the manufacture of flint containers.[10]

This plant, of course, was built after the present investigation was well under way and represents, to some extent, an independent investigation of the same problem.

Statement of Problem and Previous Work

If the glass manufacturers should insist on a medium-fine specification sand, the sand producers will find it necessary to choose one of several alternatives. These are: (1) mine only those deposits containing finer sands, (2) screen out and reject the unacceptable coarse fractions of other deposits, or (3) grind the coarse sands to glassmaker's specifications. To adopt either of the first two alternatives would leave a considerable amount of sand in the ground or in producers' stockpiles and would waste natural resources. An investigation of the third possibility led to the present study. There was no doubt that the coarse sand could be reduced to any size desired with any one of a variety of grinding machines. The problem was to do so without introducing materials, such as iron, that are deleterious to glass sand and with production of minimum amounts of extreme fines.

The problem has been under study in the laboratories of the Owens-Illinois Glass Co. for a number of years. Their efforts were on a small, batch scale, and the results were not conclusive. Grinding studies of glass sand were also conducted by other industrial laboratories but no acceptable products were made. The ground sands were either too fine, badly contaminated with iron, or both.[10]

Present Investigation

Description of Sample: The present investigation was made on a 35-ton sample of minus 20 mesh

Table III. Screen Analyses of Feed and of Ideal Glass Sand

Size, Mesh	Weight, Per Cent	
	Feed Sand	Ideal Glass Sand
Plus 35	47.8	1 - 2
35 to 48	28.9	25 - 30
48 to 80	18.4	30 - 40
80 to 100	2.8	36[a]
100 to 200	1.9	
Minus 200	0.2	
Total	100.0	100.0

[a] Minus 80-mesh material with not more than 1 to 2 pct minus 200-mesh material. The minus 80-mesh material should not exceed 40 pct. It was further stated that if it were possible to conform to the other size requirements and keep the amount of minus 80-mesh material below 30 pct, it would be desirable to do so.

washed silica sand from the Ottawa Silica Co., Ottawa, Ill. The sample was essentially silica with small amounts of pyrite, rutile, tourmaline, iron oxides, and other heavy minerals. Some of the silica grains contained inclusions of the accessory minerals; others had surface stains of iron oxide. Chemical analysis showed 0.024 to 0.033 pct total iron reported as ferric oxide. The average sieve analysis of more than 25 samples of the feed sand collected during this investigation is shown in table III.

For purposes of the present study, the Owens-Illinois Co. set up a screen analysis of an "ideal glass sand" toward which the work was directed. It was not expected that the test products would match that analysis in all respects, but it was an excellent guide. The screen analysis of the "ideal glass sand" is also shown in table III.

Grinding with Pebble Mill: To avoid contamination of the sand by iron, the first series of experiments was made with a pebble-mill grinding circuit. The mill, 15 by 36 in., was lined with silex blocks and charged with 145 lb of 1 to 2½-in. porcelain pebbles. The ends were lined with rubber belting, and the mill had a 6-in. discharge. Other items in the experimental setup included a single-spigot, hydraulic classifier, a 3-ft bowl classifier, and necessary sand pumps. In addition, an 8-in. spiral was utilized to dewater the underflow from the hydraulic classifier and convey it back to the pebble mill. A photograph of the entire unit is shown in fig. 1.

The hydraulic classifier was the subject of considerable experimentation before a satisfactorily operating device was constructed. It was patterned after an open-bottom hydraulic classifier which was developed in another Bureau of Mines laboratory and described in another publication.[1] The fact that the present problem was concerned with the classification of a single mineral permitted some simplification. It was constructed of a piece of 4-in. pipe 26-in. long fitted with a conical bottom. Feed was introduced at a point 6 in. from the top, and the hydraulic water entered 9 in. from the bottom. Fig. 2 is a photograph of the classifier. The machine was fitted with a periodic pincers which has been used on other materials to give a semicontinuous discharge of spigot product. In this case, however, it was unsatisfactory; repeated opening and closing of the pincers caused such surging of the sand pulp that classification was impossible. A few tests soon showed the most satisfactory continuous spigot discharge was through a soft rubber hose constricted by a screw clamp.

Several factors in operating a revolving mill can be varied to influence the grain size of the product, among them being mill speed, size and amount of grinding mediums, pulp dilution, rate of feed, and circulating load. Actually, in the present investigation of pebble-mill grinding, it was practical to vary only the last two. To do otherwise would have complicated the investigation unduly.

The mill speed was held constant at 44 rpm or about 65 pct of the critical speed. The volume of the pebble charge, including voids, was somewhat less than half that of the mill. The pulp in the mill was held between 55 and 60 pct solids. At less than 55 pct solids the pulp became too dilute, and at much over 60, some of the smaller pebbles floated out the mill through the feed scoop.

The test procedure was as follows: crude sand was fed to the hydraulic classifier, which was set to overflow a minus 35-mesh product. The coarse-sand underflow was pumped to the 8-in. spiral and conveyed to the pebble mill for grinding. The mill discharge was laundered to the same hydraulic classifier to close the circuit. The hydraulic-classifier overflow was pumped to the bowl classifier, where most of the extreme fines overflowed and the rake product represented the finished sand.

The circuit was controlled by taking frequent timed samples of the hydraulic-classifier overflow and wet-screening on a 35-mesh screen. Visual comparison of the quantity of sand retained on the screen with the total hydraulic-classifier overflow collected during an equal interval gave a rapid check on the operation. The hydraulic water was regulated to give an overflow containing 1 to 3 pct of plus 35-mesh sand. Amounts less than that caused excessive loss of finished fine sizes in the spigot underflow.

The results of a group of preliminary tests, showing the effect of increases in the circulating load upon the screen analysis of the finished product, are given in table IV. The circuit was first determined to have a capacity of about 2.0 lb per min at a circulating load of 90 pct. Increases in circulating load were then effected by a temporary rapid increase in feed rate until the desired sand load was attained. The feed rate was then reduced to an amount somewhat higher than the original value and the circuit balanced at the higher circulating load.

Table IV. Variation in Grain Size with Circulating Load

Test No.	1	2	3
Circulating Load, Per Cent	90	170	230
Size, Mesh	**Weight, Per Cent**		
Plus 35	1.6	1.0	2.8
35 to 48	24.0	32.0	43.7
48 to 80	41.1	42.3	36.6
80 to 100	8.4	7.6	5.8
100 to 200	16.4	12.4	8.5
Minus 200	8.5	4.7	2.6
Total	100.0	100.0	100.0

The increase in the mean grain size with increase in circulating load was very evident, particularly the rapid increase in the quantity of 35 to 48-mesh sand and the decrease in the minus 80-mesh fraction. This same phenomenon was true, of course, with variations in the rate of new feed; a decrease in the feed rate would result in a corresponding decrease in the mean grain size of the product. These two factors, rate of new feed and circulating load, were the principal means of controlling the grain size and screen analysis of the finished sand.

Before proceeding with the next group of tests, the porcelain pebbles were replaced with a charge of flint pebbles. The wear on the porcelain spheres during the preliminary tests had been severe, and it was thought that flint would be more representative of commercial practice. The new grinding charge consisted of 155 lb of ¾ to 2-in. pebbles containing approximately 35 pct each of the 2 to 1½-in. and 1½ to 1-in. size and 30 pct of the 1 to ¾-in. size.

The flint pebbles had a higher specific gravity than the porcelain grinding medium and, as expected, their use increased the capacity of the pebble mill. The first few tests with the flint pebbles soon established the fact that higher feed rates and

circulating loads than with the porcelain were necessary to produce ground sand with a satisfactory screen analysis. Table V shows the screen analyses of all products from a test in which a reasonably satisfactory ground sand was produced.

Under the conditions noted above, the power consumption and the pebble and liner wear were determined in a prolonged test during which a total of 2900 lb of sand was ground. The silex-liner loss based on per ton of sand feed amounted to 2.1 lb, the pebble consumption was 4.1 lb and the power consumed was 9.6 kw-hr.

Table V. Results of Flint Pebble-mill Grinding

a. Screen Analyses of Products

Size, Mesh	Weight, Per Cent				
	Feed	Pebble Mill Discharge	Hydraulic Classifier Underflow	Bowl Sands	Bowl Overflow
Plus 35	47.9	31.9	50.8	1.0	
35 to 48	29.1	42.0	42.6	31.4	
48 to 80	18.8	18.0	6.4	43.2	
80 to 100	2.6	2.8	0.2	9.9	
100 to 200	1.6	3.1	0.0	12.1	7.8
Minus 200	0.0	2.2	0.0	2.4	92.2
Totals	100.0	100.0	100.0	100.0	100.0

b. Operating Data and Iron Analyses of Products

Product	Amount, Lb per Min	Weight, Per Cent	Analysis, Per Cent Fe_2O_3
Feed	2.29		0.033
Pebble mill discharge	5.94		0.026
Hydraulic classifier underflow	5.90		0.027
Bowl sands	2.11	95.5	0.026
Bowl overflow	0.10	4.5	0.100

The quantities of products given in table V, part *b*, column 2, were the average of several observed values taken while the test was in progress. The circuit was easily operated and, as shown, a weight balance of the products was readily made. The simple hydraulic classifier operated with surprising efficiency. Very little sand finer than 48 mesh was returned to the pebble mill for regrinding.

The finished product (the bowl sands) compared very favorably with the screen analysis of the hypothetical, ideal glass sands. It is also of interest to note that the process was one of benefaction as well as size reduction. The iron content of the bowl sand was materially lower than that of the feed. Some of the iron oxide in the sand was liberated by the grinding and reported in the fine overflow from the bowl classifier.

The recovery of ground sand was 95.5 pct. In other words, less than 5 pct of the feed was overground and rendered useless for glass-making purposes. However, it is possible that this fine bowl overflow, containing 0.10 pct iron oxide and more than 92 pct minus 200 mesh, could be recovered for use as ground silica.

In addition to the tests outlined above, others were made in which a vibrating screen, fitted with a 35-mesh brass cloth and equipped with sprays, was used as the sizing device. This circuit gave a good product but the screen cloth had a relatively short life before it became blinded. Frequent brushing of the screen was necessary to maintain an effective separation. It is probable that other types of screen cloth, such as slotted stainless steel, would have a longer effective life but neither the time nor equipment was available for such a study.

Fig. 3—Pebble-mill and rod-ball mill.

Grinding with Rod Mill: The next phase of the investigation was a brief study of grinding the sand in a rod mill. The rod-mill shell had the same external dimensions as the pebble mill so that the two could be used interchangeably on the same driving mechanism. The rod-mill shell was lined with Ni-Hard liners, and the ends were lined with ½-in. rubber belting so that the internal dimensions of the mill were 16½ by 35 in. The mill had a 12-in. discharge to facilitate rapid exit of the ground pulp and was charged with 300 lb of rods. The rods were ¾ and 1-in. SAE 4150 steel,* which had been hardened by heat treating and oil quenching to give a surface hardness of 45 Rockwell C. Fig. 3 is a close-up showing both the rod and pebble mills.

* Contains 0.48 to 0.53 pct C, 0.75 to 1.00 pct Mn, 0.80 to 1.10 pct Cr, and 0.15 to 0.25 pct Mo, according to catalogue data.

The rod mill, of course, had a greater capacity than the pebble mill, so much more that the efficiency of the hydraulic classifier was considerably impaired. It then became necessary to conduct a series of tests during which various changes in the design of the classifier were investigated. After some experimentation the classifier shown in fig. 4 was developed and it gave good service in the subsequent work with rod and ball mills.

The new classifier utilized the basic 4-in.-diam classifying column of the other, but the top section (6-in. long) was 5 in. in diameter and the feed pulp was introduced at the bottom of the 5-in. section. The enlarged top section very effectively reduced the velocity of the entering pulp to a proper level, decreased surging, and contributed to smooth operation of the unit. The lower half of the classifier now consisted of an open-bottom, 4-in., classifying column enclosed in a 7-in. pipe with a conical bottom. The hydraulic water was introduced in the annular space between the two diameters. The function of the 7-in. pipe was to provide additional sand-storage capacity in the classifier.

Two changes were made in the operation of the circuit. The dry sand feed was introduced directly into the rod mill to improve the size distribution of the product, and the pulp consistency in the rod mill was increased to 64 to 65 pct solids. Otherwise, the flow of the pulp was the same as previously described.

To minimize oxidation in the mill and reduce the amount of iron picked up by the sand, the pulp was kept alkaline by introducing suitable reagents. The use of the alkaline pulp was not the whole solution to the problem of iron contamination, but there is no doubt it helped clean the surface of the sand grains. No attempt was made to determine the optimum quantity of reagents beyond ensuring that the pulp pH was between 9.0 and 9.6. In the rod-

Table VI. Results of Rod-mill Grinding

a. Screen Analyses of Products

Size, Mesh	Weight, Per Cent				
	Feed	Rod Mill Discharge	Hydraulic Classifier Underflow	Bowl Sands	Bowl Overflow
Plus 35	45.5	32.2	54.5	4.0	
35 to 48	29.2	38.0	36.4	33.2	
48 to 80	19.9	20.2	8.5	41.2	
80 to 100	3.4	3.6	0.5	9.6	1.5
100 to 200	2.0	4.0	0.1	11.0	14.7
Minus 200	0.0	2.0	0.0	1.0	83.8
Totals	100.0	100.0	100.0	100.0	100.0

b. Operating Data and Iron Analyses of Products

Product	Amount, Lb per Min	Weight, Per Cent	Analysis, Per Cent Fe_2O_3
Feed	4.93		0.030
Rod-mill discharge	14.56		0.050
Hydraulic-classifier underflow	10.16		0.034
Bowl sands	4.84	93.4	0.037
Bowl overflow	0.34	6.6	0.67

mill test, reported below, the reagents were sodium hydroxide and sodium silicate, 0.4 lb of the former and 1.0 lb of the latter per ton of sand feed. The results of a typical rod-mill test are given in table VI.

As shown in table VI, the new sand feed to the rod mill was 4.93 lb per min as compared to 2.29 for the pebble mill. The recovery of ground sand amounted to 93.4 pct, and 6.6 pct was discarded in the overflow from the bowl classifier. The new hydraulic classifier functioned satisfactorily, and, as before, very little minus 48-mesh sand was circulated.

The disposition of the iron was of interest as it affected the grade and value of the product. Most of the iron abraded from the grinding medium was fine enough to remain in suspension long enough to overflow the bowl classifier. The remainder increased the iron oxide content of the bowl sands to 0.037 pct as compared to the feed value 0.030 pct. Although this is not a large increase, it might be enough to render such sand unfit for certain types of flint-glass containers.

Much of the iron in the product was magnetic and could be removed magnetically. However, the addition of magnetic separation would add considerably to the cost of the process.

Fig. 4—Second hydraulic classifier.

In addition to a higher iron content, the product of the rod-mill circuit was somewhat coarse. Although it would have been possible to change the operation to produce finer sand, such changes (to increase the retention time in the mill) would tend to increase the iron content of the product also.

Grinding with Ball Mill: The final phase of the work was a study of sand grinding in a ball mill. The same flowsheet, including the grinding unit, was used, except that the rods were replaced with a charge of Moly-Cop forged-steel balls consisting of 50 lb of 2-in., 175 lb of 1½-in., and 75 lb of 1-in. balls. The same amounts of alkaline reagents were used in these tests to reduce the iron pick-up.

A few preliminary runs were made to become familiar with the action of the balls and to make necessary changes in the operating conditions. The results of a satisfactory test with the ball mill are detailed in table VII.

As shown in table VII, substitution of an equal weight of balls for rods permitted an increase in the rate of feed to 7.15 lb per min. The recovery of ground bowl sand was 90.8 pct, and 9.2 pct of the weight was in the fine bowl overflow.

The screen analysis of the bowl sands compared very favorably with that of the "ideal glass sand" and was one of the best produced during the investigation. Equally important was the fact that the iron content of this product was actually less than that of the feed. Chemical analyses for iron were made by Bureau of Mines and Owens-Illinois analysts and are shown in table VII. Although the analyses differ to some extent (due, probably, to differences in the methods of analysis), it is evident that the final bowl sand contained less iron than the feed.

Table VII. Results of Ball-mill Grinding

a. Screen Analyses of Products

Size, Mesh	Weight, Per Cent				
	Feed	Ball-mill Discharge	Hydraulic Classifier Underflow	Bowl Sands	Bowl Overflow
Plus 35	48.7	26.2	47.2	2.6	
35 to 48	28.2	33.6	36.6	24.7	
48 to 80	17.9	34.5	13.0	45.2	
80 to 100	3.5	5.7	1.9	11.4	0.5
100 to 200	1.6	6.1	1.0	13.8	12.8
Minus 200	0.1	3.9	0.3	2.3	86.7
Totals	100.0	100.0	100.0	100.0	100.0

b. Operating Data and Iron Analyses of Products

Product	Amount, Lb per Min	Weight, Per Cent	Fed. Bur. of Mines Analysis, Per Cent Fe_2O_3	Owens-Illinois Analysis, Per Cent Fe_2O_3
Feed	7.15		0.031	0.025
Ball-mill discharge	18.76		0.037	0.028
Hydraulic-classifier underflow	11.81		0.026	0.015
Bowl sands	6.38	90.8	0.026	0.020
Bowl overflow	0.65	9.2	0.40	0.42

Just why the ball mill gave a product much lower in iron than the rod mill is a matter for conjecture, but there were several contributing factors. The larger sand load carried by the ball mill probably cushioned the grinding medium, resulting in more iron-to-sand and fewer iron-to-iron contacts. The ball mill gave a finer grind; and, in so doing, the abraded iron was ground fine enough so that almost all of it overflowed the bowl classifier. Another consideration was the difference in composition and physical properties of the two iron-grinding medi-

ums. The balls were harder and may have been more abrasion resistant as well.

After determining the conditions under which a properly sized sand could be produced, a test during which 5 tons of sand were ground was made to note the wear on the balls and liners. The ball consumption was 1.1 lb per ton, and the liner wear amounted to 0.3 lb per ton of sand feed. These figures are not to be regarded as absolute determinations, the test was too short for that, but as an estimate of the iron consumption in the laboratory setup.

The power consumed during the same test was 4.2 kw-hr per ton of sand. This, of course, is an experimental value but it is likely that it could be reduced. The ball mill was not operated at maximum capacity for several reasons, but had it been possible to do so, the power per ton would have been less.

Beneficiation Tests on Fine Overflow: There are several factors, such as greater capacity per unit of mill volume and less power per ton of sand, which give the ball mill a definite advantage, in this case, over the other grinding units. The only objection which could be raised to the use of a ball mill for grinding glass sand is the amount of fines produced. In the test reported, 9.2 pct of the feed was overground and removed as the overflow in the bowl classifier. This is twice as much fines as was produced in the pebble-mill grind. There is, of course, the possibility of recovering and utilizing this fine-sand overflow, provided the iron content can be materially reduced.

To investigate this possibility, a substantial sample of the fine overflow from the ball-mill test was collected for experimental work. It was thickened, but was not permitted to dry, to keep oxidation of the contained iron at a minimum. A combination wet-screen analysis and sedimentation test was made on the material to note the size and iron distribution. The results of the test are tabulated in table VIII.

Table VIII. Screen Analysis of Bowl Overflow

Size, Mesh	Weight, Per Cent	Analysis, Per Cent Fe_2O_3	Per Cent of Total Fe_2O_3
Plus 100	0.9	0.114	0.2
100 to 200	13.4	0.033	1.0
200 to 270	17.8	0.034	1.4
270 to 325	9.7	0.061	1.4
325 to 400	11.4	0.090	2.3
Minus 400 sand	11.5	0.180	4.7
Minus 400 slime	35.3	1.110	89.0
Total	100.0	0.44	100.0

Aside from the very small amount of plus 100-mesh sand, there was a marked increase in iron content with decrease in grain size. As a result of these data, several sedimentation and classification tests were made to recover a low-iron sand product, but the results were unsatisfactory. A combination of classification and low-intensity wet magnetic separation did recover a fairly good product, but the loss of sand was high. This scheme yielded a sand product containing 0.039 pct iron oxide, with a weight recovery of 59.5 pct.

Best results were obtained by separating the entire overflow in a Ferrofilter—a high-intensity, grid-type, wet magnetic separator. By the use of this machine a 90.6 pct weight recovery was made in a fine-sand product containing 0.037 pct iron. This product should be acceptable for many applications where ground silica is specified. Among these are foundry use as mold wash, in production of glazes, and in the manufacture of such products as metal polishes, abrasive soaps, composition flooring, and paint fillers.

Summary and Conclusions

The data collected during the investigation and presented in this paper show that a graded-glass sand acceptable to the glass-container industry can be readily produced. An acceptable sand, of medium fineness, was made by grinding and double classification of a washed silica sand. The amount of overgrinding was not excessive and the finished sand was free of iron contamination.

Of those grinding units investigated, the ball mill was favored for producing a most satisfactory product at highest capacity and lowest unit power. The ball mill did produce a higher percentage of extreme fines but this fine fraction was recovered as a bowl overflow and could be beneficiated to make a usable byproduct.

This process offers sand producers a simple means, should it become necessary, of producing the medium-fine sand which the glass container industry desires. A commercial unit should be able to make further simplifications not possible in the laboratory. For instance, a multispigot hydraulic classifier could possibly be used to separate both the too-coarse and too-fine fractions from the finished sand in a single operation. In the laboratory, both hydraulic and mechanical classification were utilized to make a properly sized sand.

Acknowledgments

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The advice and guidance of R. G. Knickerbocker, Chief, Rolla Branch, Metallurgical Division, Federal Bureau of Mines, Rolla, Mo., are gratefully acknowledged.

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Production of Ammonium Sulphate and Manganese Oxides

Manganese Products, Inc. has developed a chemical process for the recovery of high-grade manganese oxides from low-grade manganese ores. Ammonium sulphate is also produced. Manganese ore is leached with sulphurous-sulphuric acids. The impurities are removed with ammonia, heat, and air. Manganese oxides are produced by addition of more ammonia and air under pressure. Ammonium sulphate is recovered from the remaining solution.

by Norman Ketzlach

"MANGANESE today overshadows all strategic materials in our planning for national security," according to a report issued recently by the American Manganese Producers Association. By far the greater part of the manganese ore used in the United States is imported.

NORMAN KETZLACH is Chief Chemist of Manganese Products Inc., Seattle, Wash.

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"We have enough manganese ore in the United States to last from one hundred to three hundred years and longer, but most of the deposits are undeveloped. As in the case of copper, lead, and zinc, manganese ore, both foreign and domestic are low grade. Manganese ores have to be concentrated and processed. After being processed, domestic ores are higher grade than foreign." Most of the manganese ore found in this country, however, is not amenable to concentration by ore-dressing methods.

Manganese Products, Inc. has developed a sulphurous-sulphuric acid leaching process for the recovery of manganese from low-grade, intermediate manganese ores. By the term "intermediate manganese ores" is meant those ores in which the manganese in the form of manganese oxides is found to have two different valences, usually two and four, as is probable in Mn_3O_4, or the ore may be a mixture of manganous and manganic oxides. "Intermediate" manganese siliceous ores are ores in which the silica is a constituent of the ore and is loosely linked to the manganese by water of crystallization.

This process has been carried out with the type of manganese ore found in the Olympic Peninsula of western Washington; Clough, N. C.; and Lava Hot Springs, Idaho, as well as just recently with manganese ore from Las Vegas, Nev.

Table I is a typical analysis of each of these ores.

Table I. Manganese Ore Analysis, Per Cent

Constituent	Wash.	N. C.	Idaho	
			Lava Hot Springs	Pocatello
Mn (total)	25.9	17.9	27.9	31.5
Mn (tetravalent)	10.9	13.5	23.7	30.3
Mn (divalent)	15.0	4.4	4.2	1.2
SiO_2	25.0		23.3	18.1
Fe_2O_3	4.95		10.5	10.0
Al_2O_3			2.0	0.5
BaO			5.1	
CaO	5.8	Trace	2.2	3.0
MgO	0.1	Trace	0.1	0.3
Ignition loss	5.8	11.1	11.2	

As far as the manganese leaching process is concerned, these ores differ primarily in the form in which the manganese oxides occur. The tetravalent manganese, the manganese existing in the dioxide form, is soluble in a sulphurous acid solution. To dissolve the divalent manganese, sulphuric acid is required.

To dissolve the type of ore found in Washington and North Carolina, a mixture of both sulphurous and sulphuric acid is used. Whereas to dissolve the type of manganese ore found in Idaho and also in Montana, sulphurous acid alone is sufficient. The Nevada manganese ore is of the same type as that found in Idaho as far as the leaching process is concerned.

It has long been known that manganese ores, with the exception of rhodonite, which is a true manganese silicate ore, can be dissolved readily in sulphurous and sulphuric acids.[1] The removal of the impurities such as silica, iron, and alumina from the resulting solution is the step that has been difficult.

The process worked out by Manganese Products has been carried through pilot-plant equipment. This process has also been extended to the production of ammonium sulphate.

Leaching of Ore: The manganese ore, ground to —65 mesh is leached with sulphurous acid and sul-

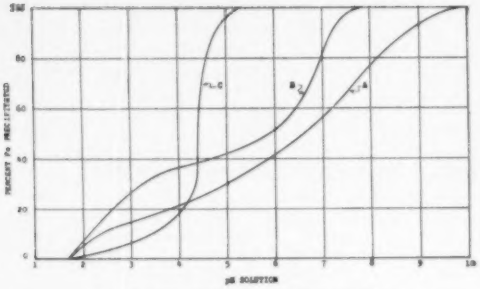

Fig. 1—Precipitation of iron vs. pH.

phuric acid at a temperature of 60°C. The sulphurous acid dissolves the quadrivalent manganese in the ore as is shown by the following equation:

$$MnO_2 + H_2SO_3 \rightarrow MnSO_4 + H_2O$$

Manganese sulphate formed according to this reaction acts as a catalyst to convert some sulphurous acid to sulphuric acid in the presence of air:

$$2H_2SO_3 + O_2 \rightarrow 2H_2SO_4$$

This sulphuric acid is capable of dissolving some of the divalent manganese:

$$MnO + H_2SO_4 \rightarrow MnSO_4 + H_2O$$

Consequently, in the treatment of ores such as found in Idaho, Montana, and Nevada, in which most of the manganese is in the dioxide form, no sulphuric acid is added to the leach.

The formation of dithionates by this leaching step must be prevented:

$$MnO_2 + 2SO_2 \rightarrow MnS_2O_6$$

Oxygen in the leaching step of the process tends to repress the formation of dithionates. By maintaining an oxygen, sulphur dioxide molal ratio of about four to one, no trouble is encountered. By this leaching operation 90 to 95 pct of the manganese in the ore is recovered.

Besides dissolving the manganese, some iron, alumina, and silica are also dissolved. The method of separation of these impurities from the manganese sulphate solution has been worked out by carefully controlling the pH, temperature, and aeration.[5]

Removal of Impurities: The precipitation of iron is shown graphically in fig. 1. Curve *A*, representing the run in which low temperature was used without the addition of air, indicates that under these conditions a pH of over 9.5 is necessary for complete precipitation. When the solution is heated, iron is completely removed at a pH of 7.6 (curve *B*), and when both heat and air are employed, the desired result is obtained at a pH of 5.2 (curve *C*). Precipitation seems to be somewhat greater in the lower pH range without aeration, but above pH 4, the air rapidly increases the degree of completion. It is apparent that heating and aeration aid in the more rapid removal of iron from the solution. It is of interest to note that all of the curves are displaced to the right with respect to theoretical curves. This is probably due to the excess SO_2 and other constituents in solution that tend to act as a buffer to repress the precipitation.

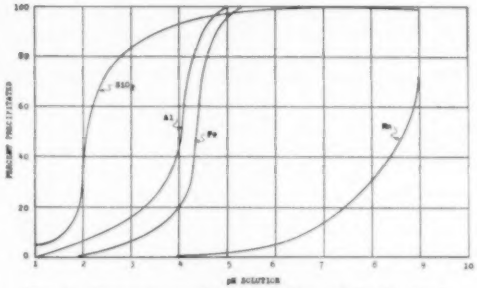

Fig. 3—Percentage of precipitation of Fe_2O_3, Al_2O_3, SiO_2, and manganese vs. pH.

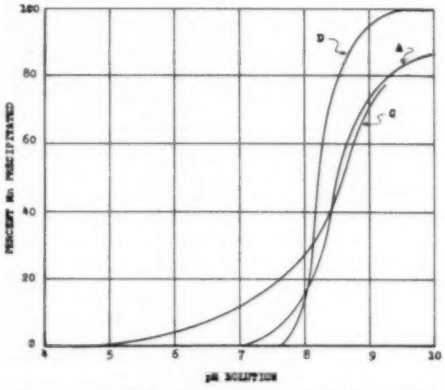

Fig. 4—Precipitation of manganese with ammonia at pH 4 to 10.

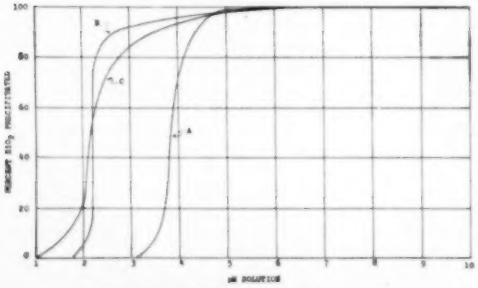

Fig. 2—Precipitation of silica vs. pH.

Fig. 2 illustrates the precipitation of silica. In the heated runs (*B* and *C*), 80 pct of the silica is removed at a pH of about 2.5, and in the room temperature run (*A*) at a pH of about 4. However, the curves reach 98 pct precipitation at pH of 5.0, and in none of them is precipitation quantitative until about pH of 7.0.

Since heating and aeration are essential, primary consideration should be given to the results which show the effect of these factors. Fig. 3 gives a plot of the percentage of Fe_2O_3, Al_2O_3, and SiO_2 precipitated, and manganese versus pH for run *C* (heat and aeration). On increasing the pH, SiO_2 precipitates first, followed by alumina, iron, and finally at a much higher pH, manganese. The action of silica

and alumina are both surprising in that they precipitate before the iron. The results show that essentially all of the impurities (iron, alumina, silica) are removed from solution at a pH of 5.2, while approximately 2 pct of the manganese is lost.

Precipitation of Manganese Oxides: The precipitation of manganese with ammonia at various pH's is shown in fig. 4. Run *A* was made at room temperature and without air oxidation. Run *B*, made with the addition of heat alone, gave essentially the same result indicated by curve *A*. Run *C* shows that aeration causes Mn to precipitate at a lower pH.

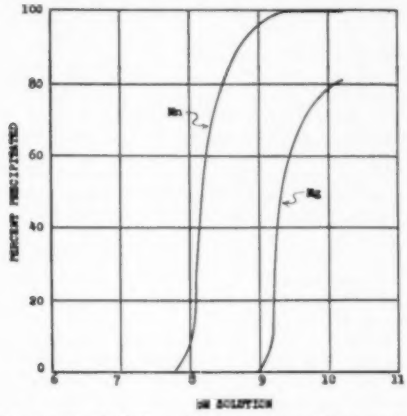

Fig. 5—Precipitation of manganese and magnesium vs. pH.

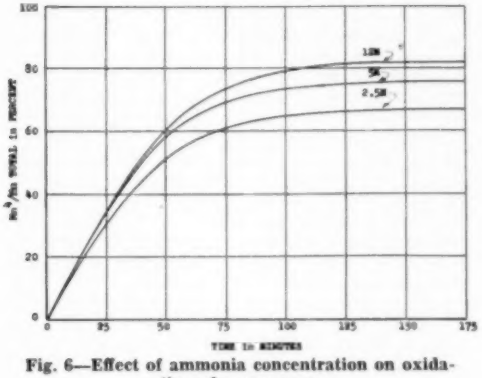

Fig. 6—Effect of ammonia concentration on oxidation of manganese.

These curves at atmospheric pressure are similar at the higher pH values. Complete precipitation of Mn without pressure requires a pH above 10.

Run *D*, at 40 psi air pressure, indicates that complete precipitation of Mn can be attained at pH 9.4, and 95 pct at pH 8.98. This run also indicates that below a pH of 8.98, no magnesium is precipitated (fig. 5). When the pH of the solution reached 9.4, 59 pct of the magnesium was removed. It is evident that precipitation conditions must be maintained so that the pH of the solution does not rise above 9.0, at which point 95 pct of the manganese is precipitated without any magnesium contamination. This care must be exercised only when manganese ore high in soluble magnesium is treated. For the purpose of these precipitation tests magnesium sulphate was added to the solution at the rate of 2 g of magnesium per liter in order to complete the information regarding the removal of impurities from the solution. However, all the ores tested were low enough in magnesium so that such careful control of magnesium contamination was not necessary.

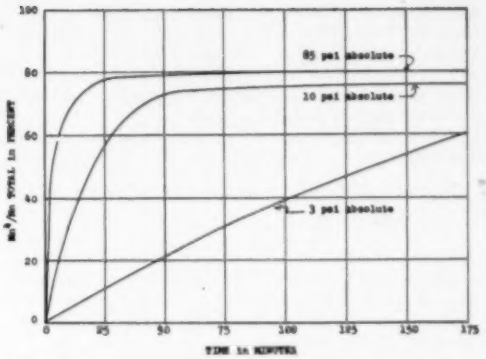

Fig. 7—Effect of oxygen pressure on oxidation of manganese.

After removal of impurities, the solution is essentially manganese sulphate with some ammonium sulphate (formed from ammonia precipitation of impurities). Under proper control a manganese oxide product, in which 70 to 80 pct of the manganese is in the dioxide form,[9] will precipitate from the manganese sulphate solution.

Fig. 6 illustrates the effect of ammonia concentration on the precipitation. The data show that for a given temperature and pressure, the degree of oxidation increases with an increase in ammonia concentration.

Fig. 7 shows that, for a given temperature and ammonia concentration, the degree of oxidation is a function of oxygen pressure, increasing with an increase in pressure.

Fig. 8 demonstrates that for a given oxygen pressure and ammonia concentration, the degree of oxidation varies inversely as the temperature.

In all of these tests an optimum of 80 pct oxidation is reached. There is evidently an equilibrium established between the tetravalent and divalent manganese. All runs were continued until complete

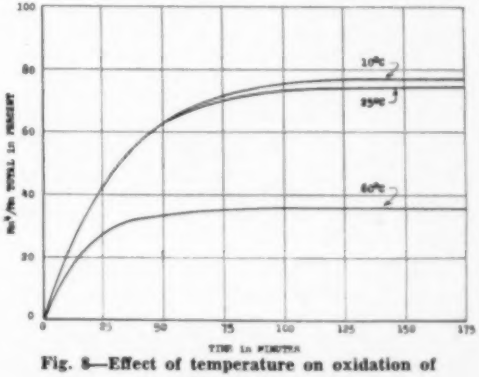

Fig. 8—Effect of temperature on oxidation of manganese.

precipitation of the manganese in solution was reached. From these curves it can be concluded that optimum conditions are reached when: (1) the ammonia concentration is above 3N, (2) the partial pressure of oxygen is above 10 psi absolute, and (3) the temperature is 25°C, or less.

The following reactions take place in this step of the process:

$$MnSO_4 + 2NH_4OH \rightarrow MnO + (NH_4)_2SO_4 + H_2O$$

$$2MnSO_4 + O_2 + 4NH_4OH \rightarrow 2MnO_2 + 2(NH_4)_2SO_4 + 2H_2O$$

The equations show that an excess of ammonia must be used to insure complete precipitation of the manganese. The following analysis is typical for the product of this process:

Component	Per Cent
MnO_2	77.9
MnO	11.5
SiO_2	0.2
R_2O_3	0.1
BaO	0.0
CaO	0.01
MgO	0.1
SO_3	0.0
Water of hydration	10.2 (approx.)

The original intention was to recover the excess ammonia by steam stripping the solution after removal of the manganese precipitate, and recovering the combined ammonia from the $(NH_4)_2SO_4$ by the addition of lime, liberating ammonia and leaving byproduct gypsum.

Ammonium Sulphate Production: The present plan, however, is to recover the ammonium sulphate. When the manganese ore is leached to a concentration in solution of 100 g Mn per liter, the equivalent ammonium sulphate formed is 24.7 lb per 100 lb of water. This is neglecting the ammonium sulphate formed from the precipitation of the impurities of iron, alumina, and silica. Ammonium sulphate can be recovered several ways: (1) addition of outside heat to evaporate the water after the excess ammonia has been stripped from the solution, (2) neutralization of the excess ammonia with sulphuric acid and addition of outside heat to evaporate the water, or (3) addition of enough sulphuric acid and ammonia and taking advantage of their heat of reaction to affect crystallization of ammonium sulphate.

Thermodynamic Considerations: In determining which method of ammonium sulphate production should be employed, markets and availability of raw materials as well as heat of reaction between ammonia and sulphuric acid must be considered. After the manganese oxides have been removed from solution, the remaining ammonium sulphate solution contains an excess of ammonium hydroxide. The addition of enough sulphuric acid to neutralize this excess ammonium hydroxide will liberate 603 Btu per lb of ammonium sulphate formed as represented by the following equation:

$$2NH_4OH + H_2SO_4 \rightarrow (NH_4)_2SO_4 + 2H_2O$$
$$-603 \text{ Btu per lb } (NH_4)_2SO_4$$

The addition of more anhydrous ammonia and sulphuric acid will liberate 894 Btu per lb of ammonium sulphate produced according to the following equation:

$$2NH_3 + H_2SO_4 \rightarrow (NH_4)_2SO_4$$
$$-894 \text{ Btu per lb } (NH_4)_2SO_4$$

Using 1000 Btu as the theoretical heat required to evaporate 1 lb of water and an evaporating efficiency of 80 pct, the material balance on the ammonium sulphate produced if the original solution is taken to dryness is:

Ammonium Sulphate Balance

Ammonium sulphate formed from

	Pct of Total
Mn precipitation	14.3
Neutralizing excess NH_3 in solution	13.5
	27.8
Additional NH_3 and H_2SO_4 added	72.2
	100.0

From the above it is seen that 27.8 pct of the ammonium sulphate is formed from the Mn precipitation. Other factors being equal, this percentage is also the approximate economic advantage of this process over that required in the direct production of ammonium sulphate from ammonia and sulphuric acid.

Taking a basis of 100 tons of 30 pct Mn ore per day and assuming 90 pct recovery of Mn, 451.8 tons is the theoretical ammonium sulphate production if all the water is evaporated.

Primarily because of the shortage of sulphuric acid at the beginning of operations, the excess ammonia will be stripped from the solution, and the water will be evaporated to crystallize ammonium sulphate.

Production Costs: In any process in which two products are produced, that portion of the costs which can not be placed definitely on one product can be divided in a number of ways. The present estimates are based on the treatment of 90,000 tons of Mn ore a year (27 pct Mn) and 90 pct recovery. This will give an annual production of 37,700 tons of manganese oxides containing 58 pct Mn and 50,700 tons of ammonium sulphate.

With the cost breakdown used, the manganese oxide production cost is $0.358 per long ton unit of Mn and the ammonium sulphate is $43.57 per ton. This manganese oxide cost does not include mining or freight on the ore. However, both costs do include amortization of an $800,000 investment in four years. The raw material costs used are based on $23.75 per ton for sulphur and $110.00 per ton for anhydrous ammonia. Also the ore for treatment is the type found in Idaho, Nevada, and Montana which does not require sulphuric acid. The ammonium sulphate cost is based on single-effect evaporation which is about 18 pct of the ammonium sulphate cost. Needless to say another saving would be made by multiple-effect evaporation.

Thus starting with low-grade, intermediate, hydrated siliceous manganese ore, found in great abundance in the United States, an economical ammonium sulphate as well as a high-grade manganese oxide can be produced.

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Some Factors in the Selection and Testing of

Concrete Aggregates for Large Structures

by Elliot P. Rexford

The quality of aggregate materials is of major importance in governing durability and permanence of concrete structures. The problem of selecting suitable aggregate materials is two-fold. Geological factors include the type and character of the aggregate source, presence of known or suspected reactive constituents, and presence of thermally incompatible materials. Laboratory investigations include petrographic studies, soundness and reactivity tests, and tests of thermal properties. The importance of each factor in the selection of satisfactory aggregate material is considered in detail.

IN the early days of concrete construction, the question of quality of sand or gravel to be used as aggregate in concrete was given little or no consideration. If the construction engineer had sufficient quantity for his structure within a reasonable distance, he was contented and gave no thought to the possibility that the material might have adverse physical, chemical, or thermal properties which could cause future deterioration of his concrete and possible failure of his structure.

ELLIOT P. REXFORD, deceased, was Chief, Petrography Section, South Pacific Division Testing Laboratory, Corps of Engineers, Department of Army, Los Angeles, Calif.

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Today a very different situation exists. With the building of larger and larger concrete structures, with a longer and longer life expectancy, the quality of the aggregate material has become of major importance. Even in small concrete structures aggregate quality cannot be passed off lightly and the quality of the aggregate must be such as to give the anticipated life without costly maintenance.

The first question to arise in the selection of an aggregate is what makes a sand or gravel suitable or unsuitable for use in concrete, and how can its quality be judged. No one rock type can be depended upon to furnish sound durable material in all instances. Granitic rocks, for example, make very satisfactory aggregates in some cases, but they are among the most unsuitable of materials when they are fractured, weathered, or highly micaceous. Limestones make excellent concrete aggregates on the one hand, and on the other they may be unsound and lack durability if they contain appreciable clay, they may have a harmful chemical reaction with the cement if they contain amorphous silica, or they may have thermal characteristics which make them highly unsuitable for use in concrete. Each rock, gravel, or sand must be proved on its own merits for the particular job under consideration.

The selection of suitable aggregate materials may be divided into two major phases, the geological phase which is carried on largely in the field, and the testing phase which of course is a laboratory problem; both are important. Very often the geological phase alone is sufficient for the selection of satisfactory aggregates for minor structures but for major structures both phases must be thoroughly investigated.

Geological Phase

In the southwest part of the country nearly every major rock type, igneous, metamorphic, and sedimentary, may be found with all imaginable intermediate phases and in all imaginable stages of alteration. Frequently very different rock types, different as to geological age, composition, and condition occur very close together. Alluvium and stream-bed sands and gravels will consist of many different rock types and even ledge rocks may vary radically as to texture and composition within the area of a quarry site. Some of the rocks make suitable aggregate materials whereas others are highly deleterious in concrete. Too often the good and the bad occur together and more than ordinary caution is required in aggregate selection.

Aggregate Sources: Concrete aggregates are usually obtained from stream-bed sands and gravels or from ledge rocks. Each has certain favorable and unfavorable characteristics and it is sometimes necessary to investigate both types to determine the most suitable source for a specific structure.

Stream Beds: The predominant sources of aggregate materials in southern California and Arizona are stream beds. Vast quantities of sands and gravels are brought from the mountain ranges by the rivers during times of flood but most of these streams are dry a large part of the time and hence these deposits make easily accessible sources of aggregate materials. From many standpoints stream-bed materials make very acceptable aggregates. The gravel pebbles and the sand grains are

frequently rounded and of satisfactory shape for a good, workable concrete mix. Often coarse and fine aggregates can be obtained together with no more complicated processing than washing and screening. "Quarrying" or excavation operations are relatively simple and the aggregates often can be obtained close to the construction site. On the other hand, stream-bed materials are derived from the various rocks of the entire drainage area of the stream in which the deposit is located. Danger of contamination by unsound and chemically reactive rocks is very high if such occur anywhere within the drainage basin. Stream-bed deposits may change in character with depth. Sometimes rock types, not observed on the surface, occur below the surface, and vice versa. Deposits having comparatively sound and durable material on the surface may contain an excessive amount of severely decomposed material only a few feet beneath it. Therefore, constant guard must be maintained to detect indications of those deleterious features in considering a stream bed as a source of concrete aggregates.

Ledge Rocks: If suitable stream-bed deposits are not available, it may be necessary to consider quarrying and processing the available ledge rocks in the vicinity of a proposed structure. Ledge rocks usually are comparatively homogeneous throughout. There is not the danger encountered in stream-bed deposits of unexpected contamination by reactive constituents or unsound weathered particles, once the character of the ledge rock is established. Ledge rocks, however, have the disadvantage of requiring relatively expensive quarrying and processing operations since crushed, unprocessed ledge rocks usually make comparatively harsh aggregate materials.

Soundness: The degree of soundness of an aggregate material is a measure of its potential durability. Unsound rock fragments, gravel pebbles, or sand grains will cause ultimate deterioration of the concrete, sometimes within a very short time. Aggregates with soft weathered surfaces or coated with soft clayey substances will have poor bonding properties in concrete and will contribute to low strength. Severely weathered, soft, micaceous or porous materials may cause localized stresses to develop in the concrete by swelling and shrinking during wetting and drying and/or freezing and thawing cycles. The ultimate result is a cracked and deteriorating structure. In the southwest where climatic conditions are not generally severe, a concrete structure made with unsound aggregate may survive somewhat longer than in another part of the country where the climate is more rigorous but it is destined to ultimate deterioration. Unsound aggregates are numerous in this vicinity and if used promiscuously, trouble may be anticipated.

Some of the rock types which may be expected to be unsound from the durability standpoint are soft, weathered, severely fractured, or highly micaceous igneous rocks, micaceous schists, soft shales, loosely bonded sandstones, and porous clayey limestones. On the other hand, hard, fresh unaltered igneous rocks (granites, diorites, diabases and gabbros), hard dense limestones, and unweathered or unfractured metamorphic rocks will generally make durable aggregate materials. Usually the potential durability or soundness of an aggregate can be judged by a consideration of the following elementary features: (1) Are the rock types dense, hard, and fresh, or porous, soft and weathered? (2) Are they fractured or severely jointed? (3) Are they coarse or fine-grained? (Generally speaking a fine-grained rock will make a more durable aggregate than a coarse-grained). (4) Do the rock types show evidence of hydrothermal or other alteration which may cause deleterious physical characteristics? (5) Do they contain a high content of mica? (6) Do they contain chemically reactive constituents which will effect an unsound condition later? (7) Are there other features such as schistosity, bedding or banding, stylolites in limestones, and so on which may influence the materials as an aggregate?

Most of these features can be observed readily in the field and on the presence or absence of such factors it can be decided from the soundness standpoint as to whether or not the material merits further consideration by laboratory tests.

Reactivity: Chemical reactivity between aggregates and certain constituents in a normal portland cement has been of increasing concern to concrete engineers for the past ten years. It is generally spoken of as alkali-aggregate reactivity since it has been found that the reactive constituents in the cement are the alkalies, sodium and potassium. These elements occur in only small amounts, from about 0.4 to 1.3 pct in all portland cements, but they are sufficient in these small amounts to cause a very undesirable condition in concrete with the wrong type of aggregate. Alkali-aggregate reactivity is known now to be one of the major causes of concrete deterioration in this, and several other parts of the country. Generally speaking this type of concrete deterioration is not well known, and its seriousness is not well understood, by many construction engineers so the subject is discussed here.

In 1940, T. E. Stanton, of the California Dept. of Highways, published an account of concrete failures along the coast of Southern California which were due to excessive expansion.[1] He found that the excessive expansion in all cases was due to a chemical reaction between the sodium and potassium in the cement and a certain mineral constituent, amorphous silica, which occurred in some of the concrete aggregates. A few months later, R. F. Blanks, of the Bureau of Reclamation, published an account of the concrete deterioration which was occurring in the then recently completed Parker Dam on the Colorado River,[2] and H. S. Meissner published a more detailed account of the Parker Dam condition in 1941.[3] Like the structures described by Mr. Stanton, the deterioration of the dam was found to be caused by a reactive constituent, a volcanic rock, in an otherwise sound aggregate. Since that time, many other concrete structures have been found to be suffering from the results of alkali-aggregate reactivity, and a tremendous amount of study and research has been performed on the subject by various federal, state, and private agencies. The results of most of these studies, together with accounts of many of the more recently discovered distressed structures, have been published in detail in engineering journals such as the issues of the American Concrete Institute, the Proceedings of the American Society for Testing Materials, and others. Therefore, anything more than a summary of alkali-aggregate reactivity as a whole, is beyond the scope of this discussion.

Probably the first question to arise in considering

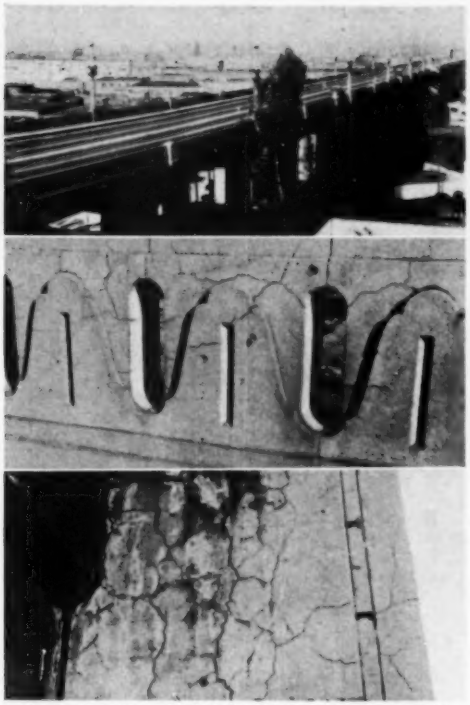

Fig. 1—Sixth Street Bridge, Los Angeles, Calif.
a. General view of bridge. b. Cracking and popouts in railing. c. Cracking in pier of bridge.

alkali-aggregate reactivity, is what constitutes a reactive aggregate? A reactive aggregate is any rock, gravel, or sand which contains one or more constituents that react chemically with the alkalies (sodium and potassium) in a normal portland cement and causes expansion of the concrete. The specific rock types found to be the most reactive are those which contain either amorphous silica (sometimes called opaline silica), or glass. Amorphous silica may occur in shales, limestones, cherts and chalcedonic fragments, while glass is frequently found in volcanic rocks. Usually the most reactive of the volcanic rocks are the more acid types, the rhyolites and andesites, but often the more basic basalts also show severe reactive tendencies.

The seriousness of distress caused by these constituents is influenced to a large degree by the amounts in which they are present in the fine and coarse aggregates. In most cases, *small* amounts of from 1 to 5 pct of reactive ingredients, in an otherwise nonreactive sand, will cause far more serious results than larger amounts. For example, many of the sands and gravels of Southern California and Arizona consist predominantly of granitic fragments with quartz and feldspar. These constituents are generally sound and entirely nonreactive. However, mixed with many of the materials is from ½ to 6 pct of opal-bearing cherts, shales and limestones, or glass-bearing rhyolites, andesites or basalts. These relatively small amounts of reactive constituents have been sufficient to cause serious reactive deterioration in road pavements, bridges, dams, and other concrete structures. In the Los Angeles area small amounts of opal-bearing shale have caused severe distress in the Sixth Street bridge over the Los Angeles River channel (fig. 1), while small amounts of volcanic particles are the cause of the well-known deterioration in the Parker, Gene Wash, Copper Basin, Coolidge, and Stewart Mountain dams in Arizona (figs. 2 and 3).

Symptoms of Reactivity: Deterioration of a concrete structure caused by alkali-aggregate reactions is made evident by several features. Popouts, pattern cracking, exudations, and expansion or swelling of the concrete are the predominant symptoms of this type of distress.

1. Popouts are conical-shaped spalls which are slowly pushed out of a concrete surface by the force of localized reaction. They may range in size from less than one inch to more than a foot in diameter. Usually they are caused by a single pebble which will be found at the apex of the cone or at the base of the socket in the parent concrete. When fresh and moist the pebble often will be soft and gel-like in texture but it will harden rapidly on exposure to the air. A rather severe popout condition has been found in the spillway and ogee structure of Sepulveda Dam near Los Angeles which has been caused by occasional fragments of amorphous silica-bearing shale (fig. 4).

2. Pattern cracks are another symptom of reactivity. Cracks of this nature may be caused by reactive constituents in either the fine or the coarse aggregate, or both. The cracks, once started, usually become longer, wider, deeper, and more numerous as time goes on until serious disintegration of the concrete has resulted. Pattern cracking may be caused by factors other than reactive aggregates,

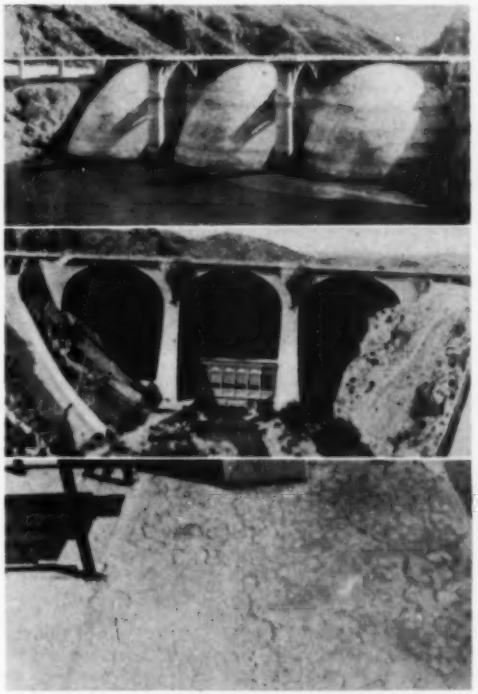

Fig. 2—Coolidge Dam, Gila River, Ariz.
a. Upstream face. b. Downstream face. c. Severe cracking in walkway on top of dam.

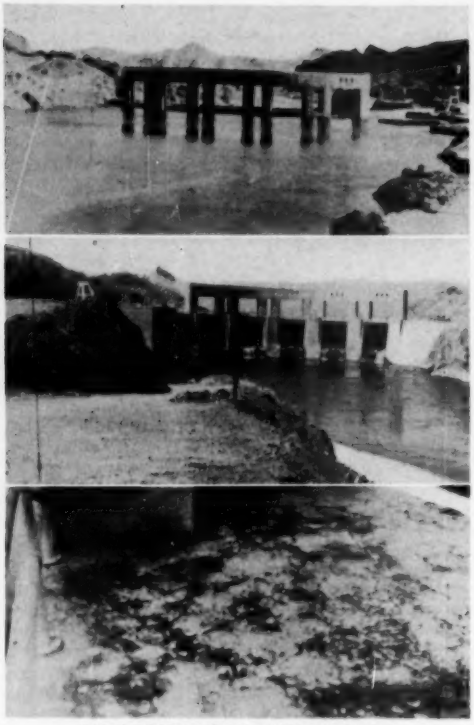

Fig. 3—Parker Dam, Colorado River.
a. Upstream face. *b.* Downstream face. *c.* Severe cracking on crest of dam.

and caution is needed when diagnosing such cracking as being due to the reactivity. Poor workmanship, overfinishing, too wet a batch, and incompatibility due to thermal properties of the aggregates will also cause this type of cracking. In the field, it is often impossible to decide which factor was the primary cause. On the other hand, a careful study of polished sections or thin sections of cores from the distressed concrete will usually indicate the probable cause. Under the microscope the thickness of a finished surface and its relation to the pattern cracks can be observed. The relation of the coarse and fine aggregate to the cracks, the presence of reactive grains and pebbles, reaction rims around aggregate particles, and gel-like reaction products can all be studied.

3. Exudations are occasionally found when the concrete is subjected to moist conditions. These exudations occur in small blobs of a jelly-like substance which exude from the concrete. When fresh and moist, it is very soft but on exposure to air the substance hardens, changes from colorless to white or brown, and generally carbonates rapidly. Chemical analyses of the substance have shown it to be a silica, or silicate gel, in which sodium, potassium, and other ions are contained. In general, the substance has a high-silica content, up to 80 pct in some cases, while the total sodium and potassium content may vary from almost negligible to as high as 26 pct or more. The presence of exudations, or gel-like rims around an aggregate particle in concrete is irrefutable evidence of alkali-aggregate reactivity (fig. 5).

4. Popouts, pattern-cracking, and exudations are surface manifestations of reactivity. Deep down in a mass of concrete, such as within a large concrete dam, reaction takes another and probably a more serious form. Here the concrete expands or swells and sometimes the hydrated cement paste loses its life and becomes chalky. Expansion is not confined to localized areas but the entire structure may grow. Arch dams may bow upstream a matter of several inches, power installations and gate structures are thrown out of alignment and long, deep cracks may occur from the stresses developed when the concrete is confined.

Chemistry of Reactivity: The actual chemical reactions which occur between reactive aggregates and the cement alkalies in forming a gel substance are not clearly understood. Several hypotheses have been suggested, but to date none has been actually proved.

One theory is that the sodium and potassium react directly with the unstable amorphous silica or volcanic glass to form a simple alkali-aggregate gel. Under this theory, once the alkalies have reacted, and have been used up in the reaction, no further distress should occur. Tending to substantiate this idea is the fact that 100 pct of highly reactive material in concrete laboratory specimens has been known to cause little or no expansion with a high-alkali cement, possibly indicating that the alkalies were used up before the concrete became rigid.

A second hypothesis suggests that the gel is the final product of a series of chain reactions and that somewhere along the series of reactions the sodium and potassium are released to react again with any available reactive aggregates. Tending to substantiate this idea is the length of time in which reaction goes on in a structure. Once started it appears to continue indefinitely, although at a retarded rate, suggesting a continuous source of alkali.

A third theory maintains that the sodium and potassium act as catalizers to start the reaction and, after starting, it is carried on by the normal calcium hydroxide of the cement paste.

A fourth theory is that the reaction once started, may involve the sodium and potassium-bearing aggregate minerals, possibly the feldspars, and thus derive a fresh supply of alkali from these otherwise nonreactive constituents. There is very little petrographic evidence to substantiate this theory, however. Feldspars and other minerals are attacked to a very limited extent by some portland cement hydrates but generally such an attack is no more than a surface phenomenon.

From these hypotheses, it can be seen that the chemistry of alkali-aggregate reactivity is still in a confused state and more work is needed on this phase of the subject.

Mechanics of Reactivity: The mechanics of the reactions, like the chemistry is not well known but a few hypotheses have been advanced of which that suggested by W. C. Hansen,[4] of the Portland Cement Association, appears to be the most reasonable. Silica can react with alkali hydroxides only in the presence of liquid water and when this reaction takes place, a relatively soluble alkali silicate is formed. Because of this solubility, the alkali silicate, theoretically, should diffuse throughout the cement paste of a concrete and cause no expansion, particularly since the alkali silicate is known to occupy less volume than that required by the silica and the alkali hydroxide. Actually, however, sili-

cate ions in solutions of alkali silicates tend to associate into complex ions which, experiments have indicated, may not be capable of diffusing through the hardened cement paste. The cement paste then acts as a semi-permeable membrane which will permit the passage of water and the alkali hydroxides but prevents the passage of the complex silicate ions. This results in the development of an osmotic pressure cell. Under such conditions, the alkali silicates formed on the surface of a reactive-aggregate pebble tend to draw water from the cement paste and form a reservoir of liquid which exerts a hydrostatic pressure against the paste. The water also brings additional alkali hydroxide into contact with the reactive pebble and additional alkali is formed. Since this alkali silicate cannot diffuse through the cement-paste membrane, it must occupy the same space as the original silica of the reactive pebble and, since the volume of the alkali silicate is greater than that of the silica from which it formed, pressure is exerted on the cement paste which is augmented by the hydrostatic pressure. Since this condition continues as long as alkali hydroxides and silica are available, tremendous pressures may result and rupture of the concrete follows. Although theoretically, crystalline silica (quartz) probably follows the same series of reactions as amorphous (opaline) silica, the rate of reaction of quartz with alkali hydroxides is so very much slower than that of amorphous silica that it is negligible in concrete.

Prevention of Reactivity: Two preventive measures for minimizing or inhibiting the deleterious activity of a moderately reactive aggregate have been suggested, both of which have been used by the Bureau of Reclamation in some recent structures.

The first, is the use of low-alkali cement. Mr. Stanton, in his original paper, suggested that reactive aggregates probably would not cause excessive expansion if used with cements having an alkali content of less than 0.6 pct. Expansion tests on laboratory mortar bars made with this type of cement and known reactive sands have shown low expansion results in most cases. It is, therefore, thought that this may be indicative of the expansion that may be expected in service. However, it is still not known whether a so-called low-alkali cement will actually prevent reactivity for all time, or will simply slow it down for perhaps a few years only to have it develop and become evident later. In this regard W. F. Abercrombia,[5] Engineer of Materials and Tests, State Highway Department, Georgia, has reported that 25 miles of road pavement in Georgia, built in 1929 with cements believed to contain 0.3 to 0.48 pct alkalies is now severely deteriorated and filled with gel-relics, and A. D. Conrow,[6] Research Engineer, Ash Grove Lime and Portland Cement Co., Chanute, Kansas, has reported having developed excessive expansion in laboratory tests with reactive aggregates when used with a cement having as low as 0.09 pct total alkali. It is his belief that the reaction once started by the sodium and potassium may be carried on by the normal calcium hydroxide of the hydrated cement. W. C. Hanna,[7] Chief Chemist, California Portland Cement Co., Colton, Calif., has shown the results of some 4-year tests of reactive aggregates with both low and high-alkali cements. His tests showed that of about 100 specimens tested the low-alkali cement bars had as much if not more expansion than the high-alkali cement bars with reactive aggregates.

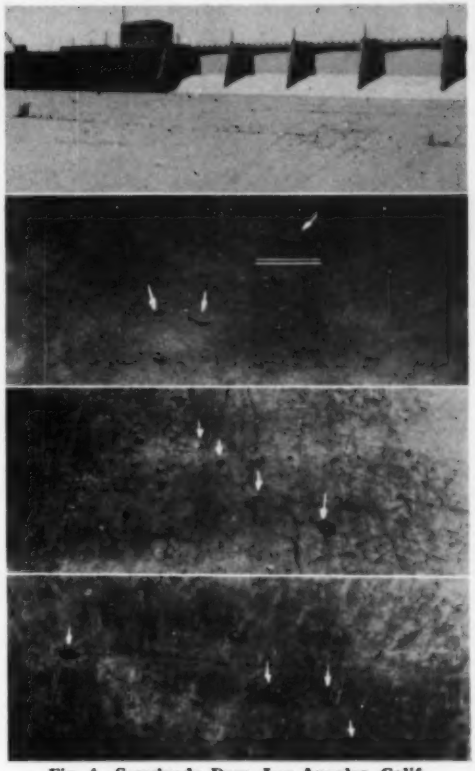

Fig. 4—Sepulveda Dam, Los Angeles, Calif.

a. Portion of spillway slab, ogee, and gate structure. b. Popouts in place in spillway slab, scale is 6 in. c. Popouts removed. Note white pebbles in cavities. d. Popouts and fine pattern cracking.

The second preventive measure is the addition to the concrete batch of an admixture of finely-divided (325 mesh), highly reactive material such as diatomaceous earth or volcanic ash. Materials of this nature have been found to inhibit or delay the activity of a reactive aggregate by, theoretically, using up or neutralizing the alkalies in the cement, before the concrete becomes rigid. W. C. Hanna,[7] has shown that finely divided calcined Monterey shale from near Los Angeles which contains diatomaceous earth (amorphous silica) was highly beneficial in retarding expansion when reactive materials were used in laboratory tests. However, again it is not known for certain whether or not the reaction is permanently prevented or only temporarily delayed. This point might be cleared up when the chemistry of the reaction is more clearly understood.

Incompatibility: Of even more recent development than the subject of alkali-aggregate reactivity, is the subject of thermal incompatibility between fine-aggregate, coarse-aggregate, and hydrated cement paste in a concrete structure. In 1941 J. A. Pearson[8] wrote on his subject, and in 1944 W. H. Johnson and W. H. Parsons[9] found, among other things, that the thermal coefficients of expansion of various rock types used commonly for aggregate materials varied widely. For example,

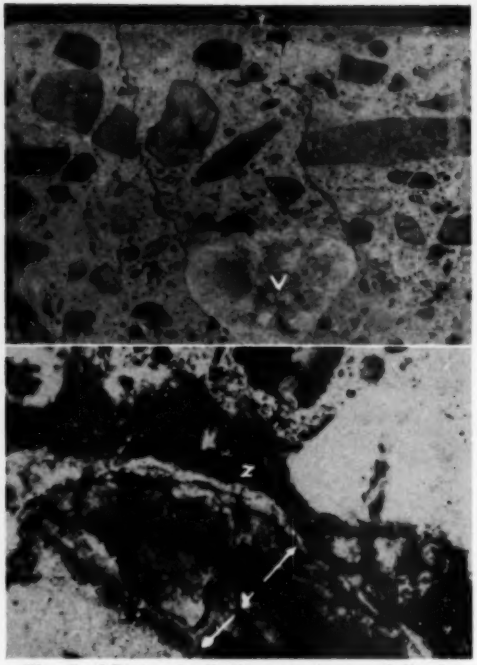

Fig. 5—Affects of alkali-aggregate reactivity in concrete.

a. Split 6-in. core. Shows cracks originating at volcanic pebble (v) passing to concrete surface (s). b. Micrograph showing dried gel reaction rim (y) between reactive pebble (lower left corner) and concrete mortar (z). Magnification 40x. (Reduced approximately one quarter in reproduction.)

the thermal coefficients of expansion of granites ranged between 1 and $5x10^{-6}$ in. per °F, limestones ranged between 0.5 and $5.1x10^{-6}$ whereas hardened portland cement paste varied between 6 and $9x10^{-6}$. Of further interest they noted that the mineral constituents of these rocks commonly found as the predominant constituents of fine aggregates, varied greatly along different axes. Quartz ranged between 4.1 and $7.5x10^{-6}$, calcite ranged between -2.8 and $14.3x10^{-6}$, and microcline feldspar was 0.6 and $9.5x10^{-6}$ for different axes. Still further, in 1947, Albert Weiner[10] showed that the thermal coefficient of diffusivity of quartz was nearly four times that of a cement mortar made with a siliceous sand. There are extreme ranges of difference in thermal characteristics and it can readily be seen that thermal compatibility between the various concrete constituents must be maintained if the concrete structure is to survive unharmed. A concrete structure is bound to crack and deteriorate with internal stresses if it contains ingredients which are expanding and contracting at difference rates and to different degrees of intensity.

Incompatibility in Nature: Probably the most spectacular examples of the results of thermal incompatibility are shown in nature. The manner in which granitic rocks disintegrate on the slopes of the desert mountains is rather strong evidence of the results of mineral incompatibility. Here the granite, consisting largely of quartz and feldspar, is soft, friable, and often completely disintegrated for from several inches to many feet below the surface. Wetting and drying, and freezing and thawing are negligible factors here, whereas heating and cooling conditions are extreme. Petrographic examinations of this material seldom show extreme evidence of chemical decomposition, that is, alterations of the feldspars to clay minerals or the iron-magnesium minerals to chlorite and so on. Generally speaking, the mineral constituents are fresh in appearance except for one feature, they are shattered and filled with incipient, tiny microscopic cracks, even in the finer sieve sizes. A light touch with a needle point will usually cause many of the sand grains, quartz, feldspar, or granitic particles to crumble, while large fragments of the granite will crumble in the hands. If we go back again to the findings of Parsons and Johnson, Weiner, and others, we have what seems to be ample proof for concluding that the disintegration of the granite, and the shattered condition of the crystals, were due to differences in thermal properties. In these areas tightly locked crystals of quartz, feldspar, and other minerals, each having widely different characteristics in expansivity and diffusivity in a rigid granite mass, must cause severe localized stresses to develop under excessive temperature changes. These can be relieved only by shattering of the mass and its individual components. If this happens in nature, it can also happen when similar materials are confined as aggregate in concrete.

Incompatibility in Concrete: In concrete, deterioration due to thermal incompatibility is generally indicated on the surface of a structure by a series of pattern-cracks similar to those found under reactive-aggregate and other conditions. The cracks, however, are generally finer and the pattern of closer texture than in the cracking due to other causes. Cracking of this nature sometimes shows up rather drastically in the desert regions, particularly if the concrete surface is wetted, but actual destruction of the desert concrete by this means alone probably does not go much beyond the pattern-cracked stage. In more rigorous climates, however, where rains and freezes are more frequent, the condition can become very serious. The fine cracks are a means for ingress of moisture and rapid deterioration will accompany any cycles of freezing and thawing.

Thus the problem of thermal incompatibility in concrete becomes very complex. The thermal characteristics of different cement hydrates are different, those of the various aggregates are all different, and even those of the various constituents in any individual aggregate pebble are different. Therefore, to insure thermal compatibility in a concrete, each major rock type from each aggregate source should be tested in the laboratory to determine its thermal characteristics, and materials having similar thermal properties must be used if future disintegration of the concrete by this means is to be prevented.

Laboratory Testing Phase

Today laboratory tests to determine the suitability of a material for concrete aggregates are numerous, and in some instances complicated and intricate, requiring expensive equipment and skilled personnel. Many of the tests such as absorption, specific gravity, compressive strength, rattler, color and sodium or magnesium sulphate tests have been used for many years and are probably familiar to anyone who has had occasion to do any concrete construction work. They are adequately described in the Standards of the American Society for Test-

ing Materials. Recently, however, outstanding tests, have been developed and are now being used by many laboratories for determining potential soundness, reactivity, and compatibility, which are not as well known and a brief description of these tests is believed warranted.

Petrographic Examination: For a complete petrographic examination of a sand, gravel or ledge rock, several types of microscopes are used ranging from a low-power stereoscope type to a high-power polarizing petrographic type. Sand grains are identified, and physical condition and quantitative estimates of composition are made by actual counts of hundreds of grains. Gravels and ledge rocks are studied in thin and powder sections under the petrographic-type microscope. The various mineral constituents are identified, and their degree of alteration is observed. Features such as incipient fracturing or incipient weathering, not visible to the naked eye, are apparent under the petrographic microscope and may indicate an unsoundness not suspected in an ordinary investigation. Chemically reactive constituents, not identified as such in the field, can be immediately detected under the microscope, even when they occur in very small amounts. The amount in which they are present can be estimated quite accurately by grain counts of a sand or by the measurements of lineal intercepts in a rock or gravel pebble. The identity of the suspected constituent can be determined and from this the susceptibility of the material to reaction can be judged. All of these features exert an influence on the ultimate suitability of the material for concrete aggregate.

Soundness Tests: With the building of larger permanent concrete structures it early became apparent that aggregates to be used in those structures must meet definite specifications for soundness and durability if the structures were to survive their anticipated life span. Tests were devised to this end among which was a slow freezing and thawing test in which the unknown aggregate was made into concrete beams and frozen and thawed at the rate of two to four cycles per day. This test soon proved to be too slow for practical purposes and an accelerated test was devised wherein the aggregate was soaked in a hot solution of sodium or magnesium sulphate, cooled, and then dried to allow crystal growth of the sodium or magnesium sulphate. Crystal growth in capillaries, fractures, or pores in sand grains or gravel pebbles caused disintegration of the material and it was judged sound or unsound by the percentage loss in weight of the aggregate after a specified number of cycles. This test is still used by many laboratories and does give information as to the potential durability of an aggregate but it also leaves much to be desired.

Very recently a modification of the old freezing and thawing test was developed[11] which is now used by all laboratories of the Corps of Engineers. It appears to be the best test for potential soundness and durability developed to date. As in the "slow" test, the aggregate is made into concrete specimens* generally 16x3x4 in. in size, but unlike the slow method the specimens are frozen and thawed automatically at the rate of twelve cycles per 24 hr. Periodically the specimens are removed from the freezer and tested electronically (sonic method) for loss in dynamic modulus of elasticity. The aggregate is considered to have failed in soundness and durability, that is to be unsuitable for use in the concrete of a large structure, if the modulus of elasticity falls below 50 pct of the original modulus of elasticity of the specimen in less than 300 cycles of fast freezing and thawing. In all tests of this kind, 3 to 9 specimens of a material are tested and the results averaged.

* Using a specially prepared cement containing an entraining agent.

Reactivity Tests: The only certain method of combating alkali-aggregate reactivity, at the present time, is to detect potentially reactive constituents, in a sand, gravel, or rock before they are incorporated in concrete, and thus prevent the use of contaminated sands, gravels, and rocks as aggregate materials. To this end, certain tests have been developed to establish the reactive nature of a selected material. These tests include petrographic examinations previously discussed, chemical tests with sodium hydroxide, and a mortar-bar expansion test. Each of these methods by itself will indicate reactive characteristics in most cases but each by itself is subject to possible error. Therefore, it is generally wise to investigate a suspected material by all the methods of test, when possible, and to interpret and correlate the results of the tests with each other.

Chemical Tests: Two chemical tests are used to indicate potential reactivity of an aggregate in conjunction with, and to substantiate, petrographic observations. The first is a sodium hydroxide test in which 10 g of crushed sample are put in a 10 pct solution of sodium hydroxide for 21 days. The solution is then analyzed for dissolved silica. With this test, it will generally be found that sands having a silica solubility of over 0.1 pct should be regarded as suspicious while those with 0.3 pct or over as potentially reactive. Interpretation of these results must always be made with caution. They must be correlated with petrographic findings or erroneous decisions may be made. The second chemical test is also a sodium hydroxide test but requires only three days to perform. It takes into account the reduction in alkalinity of the solution as well as the amount of dissolved silica. The ratio between the dissolved silica and the reduction in alkalinity of the solution will indicate the potential reactivity of the material. Again the interpretation of the results should be correlated with petrographic observations. This test was recently developed by the Bureau of Reclamation and has been described in detail.[12]

Mortar Bar Expansion Tests: The mortar-bar expansion test is the final, and probably the most acceptable, test for determining potential reactivity of a material. In this test, a graded crushed rock or sand, containing all standard sieve fractions, is mixed with a cement of known alkali content. Usually three types of cement are used, a high-alkali cement, a low-alkali cement, and when possible, the cement with which the aggregate is to be used in the structure. Mortar bars 1x1x10 in.[13] are made of the mixture and allowed to cure in a moist atmosphere in sealed containers at either room temperature (70°F) or at 100°F. Periodic length measurements are made, and the severity of reaction is estimated by the extent of the expansion in a given length of time. Usually mortar bars made of the same cements but a known nonreactive sand are tested simultaneously for comparative purposes. This test is of long duration and, although the higher curing temperature accelerates reaction, it

usually requires a minimum of six months to a year for completion. The test, therefore, loses some of its value as, generally speaking, results are needed in less than a year's time. Interpretation of the results of this test may be based on a comparison of the expansion of the unknown bars with the percent figures suggested by Blanks and Meissner.[14] They state that bars, made with high-alkali cement, which show no more than 0.02 pct expansion in six months' time may be considered as containing a nonreactive aggregate, those showing 0.04 pct expansion as suspicious, and those showing 0.1 pct expansion or more as definitely reactive.

Incompatibility Tests: At the present time, there are three tests commonly used to determine thermal incompatibility in concrete. Two different tests are used to determine coefficients of thermal expansion and one test is used to obtain coefficients of thermal diffusivity.

Thermal Expansion Tests: Thermal coefficients of expansion of aggregates, cement pastes, or concrete may be determined by either of the two following methods:

The Interferometer.[15] With this method, small specimens are ground into tetrahedrons or other suitable shapes with an overall size of about ¼ in. These are then placed in a heating unit between fused-quartz discs and viewed through an optical interferometer. A light of known wave length is used and interference rings passing the cross hairs of the interferometer are counted per unit of time as the temperature is increased and the specimens expand. Knowing the temperature range, the wave length of the light, and the number of interference rings which passed, the coefficient of expansion can be determined. This test has the marked disadvantage of using small specimens which may be hard to fabricate and hard to make representative of a heterogeneous material, such as a granite or other coarsely crystalline rock, or a cement mortar.

*SR-4 Strain Gauge.** This method has only recently been developed for use with concrete materials although it has long been used to measure stresses in steels. The gauges consist of a series of loops of very fine wire cemented between thin paper strips. They can be cemented to an aggregate specimen or to concrete mortar. The expansion and contraction of the specimen during changes of temperature causes a change in the electrical resistance of the gauge wire. This change in electrical resistance can be read directly in terms of expansion on an electronic strain indicator graduated into millionths of an inch. The gauges can be attached to a free pebble or to one confined in concrete. If the pebble is confined in concrete, a second gauge may be cemented to the adjacent mortar in order to determine the relative expansion between the pebble and its surrounding medium. The entire specimen is then subjected to temperature changes in constant humidity, and at specified temperatures the readings of the electronic indicator are observed. From these results, the coefficients of expansion of the aggregate pebble and of its adjacent mortar can be calculated and the compatibility of that pebble, or its rock type, with its surrounding matrix can be determined.

* Obtained from Baldwin Locomotive Co., Pacific Branch, San Francisco, Calif.

Thermal Diffusivity Test: To determine thermal diffusivity coefficients, a cylindrical specimen, usually a drilled core, is used. The specimen should have a length which is twice its width. A thermocouple is embedded in the center of the cylinder by drilling a small hole to the center of the piece. The specimen is then immersed in boiling water (212°F) for 2 hr, after which it is transferred to cold running water of constant temperature. The temperature difference between the interior of the specimen and the cold water is recorded at either specified differences in temperature, or at specified time intervals. The temperature differences are then plotted in degrees F against time in minutes on a semilogarithmic scale and the best possible straight line is drawn through the points. From the graph, the time elapsed between temperature difference of 80°F and temperature difference 20°F is inserted in an equation from which the thermal diffusivity can be calculated.

Conclusion

From the foregoing discussion, it may be seen that the investigation of rocks, gravels, and sands for the present-day large concrete structures may be an intricate and complex undertaking if the most suitable materials available are to be used. Full geological and laboratory investigations are time consuming and may be very costly so they are justified only when large structures are under consideration. Smaller concrete structures, however, are subject to the same factors of deterioration as the large ones and some information as to their aggregate characteristics is essential. When the structure is such that full geological and laboratory investigations are not justified, future worry and maintenance may be kept to a minimum if the proposed aggregate materials are examined in the field to determine the presence of deleterious constituents. If such are present, it would be well to have some laboratory tests made to determine the degree of their influence on concrete, or, to seek another and more suitable aggregate source.

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Executive and Finance Committees Discuss Budget and Bylaw Changes

Matters discussed by the Executive and Finance Committees of the Institute at their meeting on Jan. 20, 1950 at Institute headquarters were largely of a routine or minor nature, the more important discussion being reserved for the Board meeting on Feb. 12.

A preliminary budget for 1950 was reviewed, indicating a small balance in the black expected for the year, following four years in which expenditures exceeded income by a substantial margin. A balanced budget will be attained chiefly through an expected increase of some $55,000 in dues receipts and some $58,000 in advertising.

The Committees voted to extend to Junior Members who have been on the rolls for ten consecutive years, and who applied for a change of status to Associate Member or Member before Dec. 31, 1949, the privilege of such change of status without the payment of an initiation fee. Previously the limiting date was that of *election* rather than of *application*. Similarly, it was voted that the privilege of election as a Junior Member be extended to those who *apply* for such membership before their thirtieth birthday, instead of to those who are *elected* before they are thirty years old. A change in the bylaw to make this ruling permanent was recommended to the Board, as stated in another column. It was felt that a young man has done his part if his application was made before the time limit expired, and that he should not be responsible for the time consumed in the election procedure.

Another change proposed in the bylaws, which is expected to be acted upon at the Board meeting in April, if one is held, has to do with Affiliated Student Societies. It has been found impractical to enforce the distinction between Student Chapters, which are composed entirely of Student Associates, and other student societies composed of many students who are not Student Associates, so it is proposed in future to call all recognized student societies Student Chapters, only the officers of which would be required to be Student Associates. Another item of interest to students was the Committee's vote that all five of the national student prizes for the best papers be for $100. Heretofore one of the two graduate prizes was for $50. The student prizewinners for 1949, who will receive their awards at the Annual Meeting, are as follows, the first two being graduate prizes:

John A. Harrison, ($100) for his paper, "Progress in Mechanization in Underground Coal Mines", entered by the Chicago Section.

L. R. Kalman, ($50) for "Diffusion in Iron and Steel, a Literature Survey", also Chicago Section.

Everett O. Bracken, ($100) for "The Christensen Sillimanite Deposit Southeast of Dillon, Montana", Montana Section.

William H. Humphries, ($100) for "The Effect of Copper and Nickel Upon Rate of Sulphur Transfer", Pittsburgh Section.

Horace K. Thurber, ($100) for "Sodium Sulphate, a Possible New Utah Industry", Utah Section.

The Committee of Award was composed of Clark B. Carpenter, chairman, E. F. Schramm and William A. Staab. Entries were limited to prizewinning papers of the Local Sections that conducted contests.

A prize of $25 was authorized to be paid to Jason E. Everts, of Salt Lake City, for designing the best "key" to be made available to Student Associate members of Student Chapters.

The Committees recommended to the Board that the office of Controller be established, and that George I. Brigden, vice-president and treasurer of the St. Joseph Lead Co., be appointed to the office.

Revised bylaws for the new Mining, Geology, and Geophysics Division of the Institute were approved, subject to the approval of that Division.

Institute representatives on Engineers Joint Council for the coming year were named as follows: L. E. Young and Herbert G. Moulton, as "the most available past Presidents", with D. H. McLaughlin and Edward H. Robie, ex officio. It was noted that Dr. Young would be president of EJC for the coming year and Mr. Robie secretary.

Various AIME representatives were named as follows: Harvey S. Mudd reappointed to the Rocky Mountain Income Committee for three years. C. B. Sawyer to succeed C. H. Mathewson on the council of the American Association for the Advancement of Science. Paul Weir at the presentation of the Washington Award for 1950 to Wilfred Sykes on Feb. 27. Scott Turner to succeed himself on the Hoover Medal Board of Award for six years.

The *Drift* of *Things*

as followed by *Edward H. Robie*

Joe Alford Succeeds Bill Strang

Following the resignation of William H. Strang as executive secretary of the Petroleum Branch of the Institute, the problem immediately arose of securing a successor. Many were suggested for the job, but after the field had been carefully considered one name stood out as the most acceptable candidate—Joe B. Alford, who, since its founding a year ago, has been editor of the Journal of Petroleum Technology. His appointment was unanimously approved by both the outgoing and incoming boards of the Petroleum Branch. On general principles we are highly in favor of the promotion of editors to better jobs; specifically, we have every expectation that Joe will meet all his new responsibilities most acceptably. The worst angle to his appointment was that it left JPT without an editor but as this is being written we understand that that position is in a fair way to being filled.

Joe is a native Texan and was graduated from Texas Technological College at Lubbock with a degree in business administration, which training will perhaps stand him in better stead in many aspects of his new work than if he had chosen petroleum engineering. He entered the Army in 1940 and rose from the bottom, being commissioned in 1942 and released with the rank of Major in 1946. He then went to Stanford to get an M.A. in Journalism, returning to Dallas in 1947 for a job in that field. His attractive wife, whom he married in 1942, will, incidentally, be no handicap to him in his new work. They have a two-year-old daughter.

Local Section Programs

Study groups, or discussion groups, to exchange opinions on subjects of specific professional interest have been most successful additions to the programs of some Local Sections, particularly in the Petroleum Branch. Those Local Sections that have not tried such meetings might well consider their possibilities.

One such group is being sponsored by the Gulf Coast Section of the Institute. A. G. Gueymard, Secretary-Treasurer, sends us the following brief description of its work: "At the present time we have established one discussion group, which is covering the subject of petroleum reservoir engineering. It is highly possible that other groups will be started in the future to study other phases of petroleum engineering. The present group meets once each month for a period of two to three hours. For each particular meeting a certain phase of reservoir engineering is set up for discussion, and one or more experts on this phase are available to conduct a discussion. Usually the speaker for the evening gives a general talk on the subject and then the floor is open for discussion by all concerned. The length of these meetings depends entirely upon the desires of the members present.

"The discussion group is particularly appealing to younger members of the AIME who are interested in gaining the knowledge and ideas of others on the subjects of their interests. We think the discussions are most beneficial and worthy of the amount of work necessary to keep the group going. The one drawback to this type of study group is that the discussion often becomes too elementary and detailed. Of course this undesirable situation must be prevented if possible by the directors of the group. The discussion group meeting is entirely separate from the regular monthly meeting of the Section; however, all AIME members are invited to attend."

Constant Turmoil

Writing items for this department that have anything to do with the news of the day has become increasingly difficult under the new publishing program, for what we have to say this sixth day of February will not be in the hands of some readers until the latter part of March. It seems fairly safe to say, however, that whenever this is printed a coal strike will either have just been settled, or one will be on, or one will be threatened, so a few remarks on the subject will not completely lack timeliness.

Undoubtedly John L. Lewis has done a lot to secure higher wage rates and shorter hours for the coal miners since he first began to take an interest in them. Also undoubtedly, conditions under which large numbers of them have had to live have been bad, their work hard and dangerous, and their comparatively high wage rates misleading, for most of them do not know what it is to have a steady job five or six days a week throughout the year. And yet, sympathize though we do with them, we do not think the methods used by Mr. Lewis for their betterment are having the results intended. Over the years the increased cost of producing coal has forced prices so high that a good share of the former market has been lost to oil and gas and even to hydroelectric power. Another factor that has influenced some consumers to turn to other fuels is the always present threat of supplies being cut off by a strike, a threat which rarely exists among producers of competitive fuels.

In fact, the market for coal has dwindled to such an extent that Mr. Lewis recently invoked the three-day week in an effort to prevent an undue accumulation of stocks. Now even at the high prevailing wage rates, miners cannot get along on the income from three days of work a week, and many of them have therefore been inclined to stay away from work entirely until a full week's work is available to them. Instead of doing something constructive to provide this work, Mr. Lewis continues to harp on ever-increasing wages, the present demand being for $15 a day instead of $14.05, and an increasing royalty to take care of their old age, which, if they had a good annual income, they should be able to provide for themselves out of earnings. It has been stated that it will take the average miner six to seven years at the 95¢ increase in the daily wage to reimburse him for the time he has lost on strike during the last year.

As the wage rate and royalty payments go up, so will the price of coal, which will result in less of it being used, which will mean less work for the miners, which will mean they will have to have higher wages for the time they *do* work, which is where we came in.

Now there is no particular point in talking about these things unless we have something constructive to suggest, so we hereby offer to the coal miners what we will modestly call the Robie Plan: Determine the average number of hours worked by the coal miners per year for, say, the last two years, and multiply this by the average wage rate, to get the average yearly earnings. Add a fixed amount, say 20% to this total, and divide the sum by the number of hours that would be worked if the miners had the opportunity of working forty hours a week for fifty weeks in the year. This would give a new average wage at which the miners would go to work. It would be far below the wage they are getting now—when they get it—but they would have steady work and their annual earnings would be higher by the 20%. The cost of coal mining would drop considerably, and the price as well, so consumption should immediately begin to go up. It might take a little time for the market to absorb all that would be produced through a continuous forty-hour week, but the Government is so used to buying potatoes, and eggs, and fruit, and grain and other things that won't keep, that it shouldn't mind scattering a few stock piles of coal around the country. From cheap coal, gas and synthetic gasoline might be made, competitive with the products of the petroleum industry.

So there's the blue print. Those who know more about the coal industry than we do can work out the details.

Better Than the Atom Bomb

No sooner had the atom bomb, horrible as it was, became accepted in our minds as a possible back-to-the-wall resource in case of war than the hydrogen bomb suddenly appeared on the scene. This is supposed to be a thousand times as destructive as the A-bomb—just like a little chunk of the sun tossed on the earth, so we gather. Rightly, we think, it was decided that we should make it, but it will be quite another question as to whether we should ever use it. Certainly not use it first. And yet, if some one else uses it first maybe we wouldn't be here to use it second. In any case, whether we have the H-bomb or just a passel of A-bombs, the prospect of war is now too horrible to ponder. Certainly some small percentage at least of what we are spending on military things should be spent on propaganda and methods of assuring peace. Armament races, throughout history, have led to war. It is a fond delusion that by making every effort to keep ahead of our competitors we can achieve "security"—a much overworked and misleading word.

It is heartening to see, therefore, in newspapers like *The New York Times* and the *New York Herald Tribune*, that columns of space are being given to proposals to secure world peace disarmament. Senator Brien McMahon, the highest Congressional authority on atomic energy, made a thrilling speech on the subject in the Senate on Feb. 3. A month before, the House Foreign Affairs Committee unanimously voted to act on the world federation resolution HCR 64. Nineteen Senators, headed by Charles W. Tobey, are behind a resolution declaring it to be the sense of Congress that the United States should promote the strengthening of the United Nations and "seek its development into a world federation open to all nations with defined and limited powers adequate to preserve peace and prevent aggression through the enactment, interpretation, and enforcement of world law."

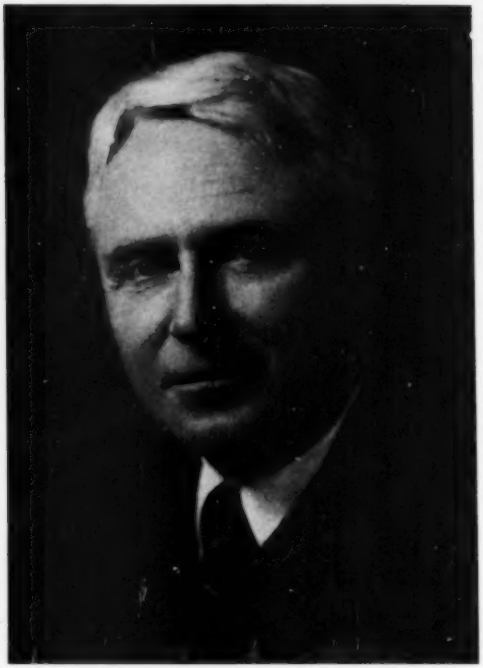
At a hearing in Washington on Feb. 3 Senator Wayne Morse described world federation as a "practical step we can take now in our search for lasting peace." Senator Claude Pepper said "the United Nations, through a world federation of nations, must be given powers to prevent war; power to enact, interpret, and enforce law."

Since publication of "The Anatomy of Peace" a few years ago, on which we commented at length in these columns at the time, several other books have developed the subject further, among the most recent ones being "Peace or Anarchy" by Cord Meyer Jr., and "The World Must Be Governed," by Vernon Nash. The several organizations promoting world peace finally united their activities in the United World Federalists, 7 E. 12th St., New York City 3, which for $1.50 a year publishes a monthly report of progress toward world federation entitled "World Government News." The idea has been spreading fast, and thousands of additional adherents have been stimulated by the prospect of the H-bomb. If a movement may be judged by the character of its enemies, it might be added that Soviet Russia, the Hearst press, and the *Chicago Tribune* are violently opposed.

World federation and world law may not be the answer, but nothing so far advanced seems as promising. If we can spend fifteen billion dollars a year on armaments, certainly we can afford a billion or so to develop some more civilized and likely plan of securing our future happiness and prosperity than making the H-bomb.

Donald H. McLaughlin

AIME President for 1950



To the mining man Homestake and Cerro de Pasco are as well known as Chevrolet and Ford are to the car industry; the mining man connects certain names with the history and development of these two companies as readily as the industry recalls the feats of the various models. One name closely associated with the more recent history of both companies is that of Donald Hamilton McLaughlin, who is President of the AIME for 1950.

It's a pretty well known fact that Dr. McLaughlin became head of the Cerro de Pasco Copper Corporation's geological department at an unusually youthful age—he hadn't completed his twenties. It's also common knowledge that Cerro de Pasco's geological department became one of the largest of any mining company, so he had a full time job supervising exploration and development programs of the corporation's extensive copper, silver and coal deposits in various parts of Peru. As one of the first American engineers to do extensive work in South America, he ranks as a pioneer in a real Pan American movement.

When he went to Harvard in 1925 as professor of mining engineering and geology, he became one of the youngest professors in the history of the university. (Some dates might help here—Dr. McLaughlin was born in San Francisco on Dec. 15, 1891, graduated from the University of California in 1914, received his doctor's degree in geology from Harvard in 1917, and from 1917 to 1919 was addressed as Lieutenant in the 63rd Infantry, U.S. Army; he went to Peru in 1920.) While at Harvard he became chairman of the Division of Geological Sciences and of the Department of Geology and Geography. In addition to his academic duties he maintained his interest in the field as a consultant, his principal client the Homestake Mining Co. This company was incorporated in California a good many years before Dr. McLaughlin put in his appearance there, but by the time he had acquired a consultant's background he was able to help solve certain geological problems at Homestake, of considerable value both in control of grade as well as in the development of additional reserves of ore. In 1945, after many years as consultant and a director, he became president of the company.

The lure of California called him in 1941 and back he went to become dean of the College of Engineering at his alma mater, the University of California. Big business claimed him two years later when he was made vice-president and director of Cerro de Pasco. Sandwiched in along the way, he has been chairman of the Advisory Committee on Raw Materials of the Atomic Energy Commission; chairman of the National Minerals Advisory Council; and a member of the task force on natural resources of the Hoover Commission. He is a director of the American Trust Co., the Empire Trust Co., the Bunker Hill & Sullivan Mining and Concentrating Co., The Dorr Co., the International Nickel Co. of Canada, San Luis Mining Co., and the Triumph Mining Co.

There has also been time for him to be chairman of the first AIME Committee on Geophysics and to have a hand in getting out the outstanding Geophysical Prospecting volume in 1929; he has served as chairman of the Mining Geology Committee and as a member of the Papers and Publications Committee, and was a member of the Committee on the Lindgren Volume. In 1947 he was a vice-president of the AIME.

Among the Student Associates—

The Mineral Technology Association of the **University of California** at Berkeley brought last semester to a close with a top notch banquet at which A. J. Carlson, chairman of the mineral technology division of the College of Engineering, was toastmaster. Arnold L. Brown, retiring president, installed the new officers for the spring semester: William W. Carder, President; Myron R. Elliott, Vice-President; Fred R. Mulker, Secretary; Alan V. Levy, Treasurer; David M. Hanger, Goat; Kenneth V. Zerda and William Davis, representatives to Engineers' Council; and Bernard York, Faculty Sponsor. A full program of activities has been planned for the semester, including four regular meetings, an open house, party, picnic, and banquet. It sounds interesting. . . . America's urgent need for trained professional men in mining, metallurgy, and geology was the keynote of the Drill and Crucible Club banquet, **South Dakota School of Mines and Technology**, on Jan. 14. Celebrating its 30th year as a student chapter, the club is the oldest organization on campus. Arthur I. Johnson, consulting engineer of Keystone, S. Dak., who was one of the officers of the club when it was organized in 1920, spoke of the great need for mining, metallurgical, and geological engineers in this country, whose future welfare depends upon the proper development of mineral resources. Warren E. Wilson, president of the School of Mines, urged the men "to direct energies seriously to prepare for their profession." J. Paul Gries, of the geology department, announced cash prizes in the AIME national contest for papers dealing with aspects of mining development and processing. Dr. Gries, who is chairman of the Black Hills Section, **AIME**, pointed out that the club precedes its official parent by about 16 years. Guy V. Bennett, president of the club, had the job of introducing the speakers.

What Went on at Recent Local Section Meetings

SECTION	DATE	ATTEND-ANCE	SPEAKER, AFFILIATION, AND SUBJECT
Ajo Subsection	Jan. 12	28	H. E. Krumlauf, University of Arizona, on copper mining in Michigan.
Boston	Jan. 9	49	C. D. Kellogg, New England Council, on New England's opportunity in steel.
Boston	Feb. 6	29	W. H. Coburn, W. H. Coburn & Co., on investments.
Carlsbad Potash	Jan. 18	51	J. E. Allen, New Mexico School of Mines, on "Monsters of the Past."
Chicago	Feb. 1	256	H. W. Johnson, Inland Steel Co, and C. D. King, U. S. Steel Corp., on prospective developments in steel making during the next decade.
Colorado	Jan. 24	33	J. H. Johnson, Colorado School of Mines, on "A Geologist Visits Guam."
Detroit	Jan. 16	85	C. Altemburger, Great Lakes Steel Corp., on plastic flow in metals.
East Texas	Jan. 10	45	R. H. Rantala, Pure Oil Co., on the Van oil field after twenty years.
Gulf Coast	Jan. 10	224	Annual banquet in honor of L. E. Young, President, AIME.
Manila	Jan.	—	C. Weekly, Marsman & Co., on Marsman plans for resuming mining.
Montana	Jan. 30	45	R. G. Bowman, Anaconda Copper Mining Co., on China's mining districts.
New York	Jan. 12	155	P. B. Counselman, the Derr Co., on roasting and calcining by fluosolids.
Ohio Valley	Jan. 19	39	L. E. Young, President, AIME, on social and political situation in England, Australia, and New Zealand.
St. Louis	Jan. 20	49	A. B. Cleaves, Washington University, on geological engineering of the Pennsylvania Turnpike.
San Francisco	Jan. 11	48	Movie, "No Man Is An Island."
Southwestern N. Mexico	Jan. 10	51	Annual Meeting
Utah	Jan. 19	87	E. W. Steinbach, and T. W. Saylor, International Smelting & Refining Co., on Tooele slag treatment and deleading plant, and converting of dross reverberatory speiss and matte at Tooele, respectively.
Permian Basin	Jan. 23	143	B. R. Weaver, Tide Water Associated Oil Co., on acidization in the Permian Basin.

St. Louis University Student Chapter members on their visit to the St. Joseph lead mines, St. Francois, Mo., last fall spent the day underground.

AIME Personals

John C. Anderson, Jr., can be reached at the Kaiser Aluminum and Chemical Corp., Mandeville, Jamaica, B.W.I. He had been in Permanente, Calif., with the Permanente Metals Corp.

Richard C. Anderson, after three and a half years as mine engineer for the Magma Copper Co., Superior, Ariz., resigned in order to attend the American Institute for Foreign Trade. At this Institute he is preparing for work in mining in Latin America.

John C. Cosgrove

John C. Cosgrove, consulting engineer of Johnstown, Pa., accompanied by Mrs. Cosgrove and their daughter Betty, is spending the winter at their residence, 845 Bay Esplanade, Clearwater, Fla., where they expect to remain until the first of May. Mr. Cosgrove was recently elected to the board of directors of the Arts and Science Club of Clearwater and has also been elected a member of the Clearwater Yacht Club.

Karl V. Lindell

E. R. Goter has moved from Roseland, Va., to Tahawus, N. Y., where he has taken a job as junior metallurgist with the National Lead Co. The operations of the American Rutile Corp., by whom he was employed, have been permanently suspended and the corporation dissolved.

J. George Grunenfelder, of the zinc plant at the Great Falls Reduction Works, became professor of metallurgy at the Montana School of Mines last fall, succeeding **John R. Spielman,** who joined the staff of the State College of Washington to become dean of the School of Mines.

Leon B. Haigler graduated from North Carolina State College last June and is now working for the Blue Diamond Coal Co. at Leatherwood, Ky.

O. W. Hakala is a junior engineer with the Hecla Mining Co. at Burke, Idaho.

A. Finley Harper has been appointed chief, division of safety and inspection, department of industrial relations for the State of Alabama. Headquarters of the division are located in Birmingham. Prior to this appointment, Mr. Harper was superintendent of mines for the Consolidated Coal Co.

Warren F. Harter is plans engineer with the Missouri State Highway Department Mail reaches him at 604 W. Main St., Flat River, Mo.

K. C. G. Heath, after three years of independent practice as a consulting mining engineer, has joined the London staff of the Anglo American Corp. of South Africa, 11 Old Jewry, London, E.C. 2.

George K. Foster has been promoted to production manager of the asbestos fibre division of the Canadian Johns-Manville Co., with headquarters in Montreal. For the past four years he had been mine manager of the company's Jeffrey mine at Asbestos. **Karl V. Lindell,** who had been superintendent of underground operations and plant engineer at the Jeffrey mine, has been made manager of the Jeffrey mine.

John C. Kinnear, Jr., has been promoted to assistant general manager of the Nevada division of the Kennecott Copper Corp.

James E. Kuykendall can be reached in care of the New York and Honduras Rosario Mining Co., San Juancito, Honduras.

E. C. Lane, formerly with the New York and Honduras Rosario Mining Co., is employed now as a junior mining engineer with Pato Consolidated Gold Dredging, Ltd., Pato, Antioquia, Colombia.

Cecil E. Bales

Cecil E. Bales has been elected president of the Ironton Fire Brick Co., Ironton, Ohio. He was formerly vice-president of the company and has been active in the technical development of metallurgical refractories for over thirty years. A member of the AIME since 1936, he has served as chairman of the ceramic materials committee of the Industrial Minerals Division and is a member of the executive committee of the Ohio Valley Section.

O. Paul Lance announces that the firm of Lance and Mitchell has been dissolved. He will continue under the name of O. Paul Lance, Consulting Engineer, engaging in general, civil, and mining engineering practice, at 1021½ E. Missouri, El Paso.

George K. Foster

T. J. Jensen

T. J. Jensen has been made sales manager of the Colorado Iron Works Co. Before joining Colorado Iron Works in 1943 as field engineer, Mr. Jensen worked for mining companies in the States, Alaska, and South America.

A. H. Miller, consulting engineer of New York City, is doing an extensive examination and appraisal job in western Canada, western United States and Mexico.

Reuben L. Olson and his wife celebrated New Year's Day with the arrival of their daughter, Sheila Anne. Mr. Olson is a mining engineer with the Oliver Iron Mining Co.

L. G. Immonen has been appointed concentrator superintendent of the Kennecott Copper Corporation's Nevada mines division, McGill, Nev. He succeeds **F. M. Jardine, retired.** Mr. Immonen received his B.S., E.M., and M.E. degrees from Michigan College of Mines.

L. G. Immonen

COMING EVENTS

MARCH

1 **Chicago Section, AIME. F. N. Rhines on recent developments in powder metallurgy.**
3 **Columbia Section, AIME.**
6 **Boston Section, AIME.**
6 **Pittsburgh Section, AIME.**
7 **Washington, D. C., Section, AIME.**
7 Society for Applied Spectroscopy, New York City.
8 **Chicago Section, Open Hearth Committee, Iron and Steel Division, all-day meeting, Del-prado Hotel, Chicago.**
8 Connecticut Section, AIME, visit to Detroit Steel Corp.
8 **El Paso Metals Section, AIME.**
8 **San Francisco Section, AIME.**
8-10 API, Adolphus Hotel, Dallas.
9-11 National Society of Professional Engineers, Nashville, Tenn.
10 **Rio de Janeiro Section, AIME.**
14 **Delta Section, AIME.**
14 **East Texas Section, AIME.**
15 **Southwest Texas Section, AIME.**
15 **Western Section, Open Hearth Committee, Iron and Steel Division, Roger Young Auditorium, Los Angeles.**
16 **Carlsbad Potash Section, AIME.**
16 **Utah Section, AIME.**
17 **Oregon Section, AIME.**
20 **Detroit Section, AIME.**
21 **Gulf Coast Section, AIME.**
23-25 Dept. of Geology, Univ. of Kentucky, and Kentucky Geological Survey, symposium on minerals in southeastern U. S., Univ. of Kentucky, Lexington.
27 **Alaska Section, AIME.**
28 **Montana Section, AIME.**
28 ASTM, St. Louis district, St. Louis, Mo.
29-31 API, Skirvin Hotel, Oklahoma City, Okla.

APRIL

3 **Boston Section, AIME.**
3 **Pittsburgh Section, AIME.**
4 **Washington, D. C., Section, AIME.**
4-7 Nat'l. Assn. of Corrosion Engineers, St. Louis.
5 **Chicago Section, AIME. T. L. Joseph on problems in iron smelting.**
7 **Columbia Section, AIME.**
10-12 **Open Hearth Conference, and Blast Furnace, Coke Oven and Raw Materials Conference, Netherland Plaza Hotel, Cincinnati.**
11 **Delta Section, AIME.**
11 **East Texas Section, AIME.**
12 **El Paso Metals Section, AIME.**
12 **San Francisco Section, AIME.**
12 **Southwestern New Mexico Section, AIME.**
14 **Rio de Janeiro Section, AIME.**
19-21 ASCE, spring meeting, Los Angeles.
21-22 **New England Regional Conference, Institute of Metals Division, AIME, Hotel Sheraton-Biltmore, Providence, R. I.**
23-26 American Ceramic Society, annual meeting, New York.
24-26 AMC Coal Convention, Netherland Plaza Hotel, Cincinnati.
25-26 Annual Metal Powder Show, Book-Cadillac Hotel, Detroit.

MAY

24-25 American Iron and Steel Institute, Waldorf-Astoria Hotel, New York City.

JUNE

6-17 Second Mechanical Handling Exhibition and Convention, Olympia, London.
8-10 National Society of Professional Engineers, Boston, Mass.
24-30 First international conference on coal preparation under direction of Charbonnage du France.
26-30 ASTM 53rd annual meeting and 9th exhibit of testing apparatus and equipment, Atlantic City, N. J.

JULY

3-8 Second Conference on Oil Shale and Cannel Coal, Glasgow.
10-15 World Power Conference, London.
30-31 Mining Association of Montana, Hotel Finlen, Butte.

AUGUST

28-31 American Mining Congress, metal mining convention and exposition, Fair Grounds, Salt Lake City.

SEPTEMBER

19-23 Conference on Wire Ropes in Mines, Ashorne Hill, Warwickshire, England.

OCTOBER

4-5 Texas Mid-Continent Oil & Gas Assn., annual meeting, Dallas.
4-6 **Petroleum Branch, AIME, New Orleans.**
9-13 **Washington, D. C., Section & Mineral Economics Division, AIME, joint meeting, Shoreham, Washington.**
12-13 **Petroleum Division, AIME, Elks Club, Los Angeles.**
13 **Southwestern Section, Open Hearth Committee, Iron and Steel Division, Houston, Texas.**
20 **Eastern Section, Open Hearth Committee, Iron and Steel Division, annual all-day fall meeting, Warwick Hotel, Philadelphia.**
21-27 National Metal Congress, Chicago.
23-25 **Coal Division, AIME, Fuels Division, ASME, Hotel Statler, Cleveland.**
23-25 **Institute of Metals Division, AIME, fall meeting, Hotel Sheraton, Chicago.**
27 **Southern Ohio Section, Open Hearth Committee, Iron and Steel Division, annual all-day meeting, Deshler-Wallick Hotel, Columbus.**

NOVEMBER

3 **Pittsburgh Section, Open Hearth Committee, Iron and Steel Division and Pittsburgh Section, AIME, annual all-day meeting, William Penn Hotel, Pittsburgh.**
9 AMC, coal division conference, William Penn Hotel, Pittsburgh.
16-18 GSA, annual meeting, Hotel Statler, Washington, D. C.

DECEMBER

7-9 **Electric Furnace Steel Conference, Iron and Steel Division, Hotel William Penn, Pittsburgh.**

FEBRUARY 1951

19-22 **AIME, annual meeting, Jefferson Hotel, St. Louis.**

APRIL 1951

2-4 **Open Hearth and Blast Furnace, Coke Ovens and Raw Materials Conference, Iron and Steel Division, Statler Hotel, Cleveland.**

James A. Neustaedter, back in this country again after working for Surinaamsche Bauxite Maatschappij in Paramaribo, Surinam, now with the Alcoa Mining Co., can be reached at 42 E. Webster Ave., Roselle Park, N. J.

James H. Taylor

James H. Taylor, after a number of years with various mining companies in the States, Philippines, Peru, Ecuador, and Bolivia, has retired to the United States and may be heard from care of H. T. Kille, Box 2063, Hollywood, Fla.

Thomas B. Reifsnyder, a recent graduate of the Colorado School of Mines, has accepted the post of sales engineer with the International Machinery Co. and is addressed in care of Grace y Cia., Santiago, Chile.

Paul L. Russell is working for the Bureau of Mines, 43 Anvil Points, Rifle, Colo.

Herman Stigzelius, chief of the State Bureau of Mines, Trade and Industry Ministry, Helsingfors, Finland, visited mining districts in the States and in Canada from September to January and wishes to express his thanks to everyone who helped him while he was here.

Edwin A. Sweetman, mining engineer of Green Bay, Wis., made a business visit in January to Kansas City and Minneapolis. While in Minneapolis he attended a meeting of owners of the Nicollet Hotel.

Ernest E. Thurlow has been transferred recently from the New York office of the raw materials operations, exploration branch of the U. S. Atomic Energy Commission to a new extension of that office in Spokane, Wash.

J. G. Van Blom is with the Domaniale Coal Mining Co., Domaniale Myn-My, Kerkrade, Netherlands.

Paul Semko is working in the geological investigations division of O. J. Porter & Co., consulting engineers, 415 Frelinghuysen Ave., Newark 5, N. J.

Robert H. Shepard has taken the post of geologist with Southern Petroleum Exploration, Roswell, N. Mex. Up to last Dec. 15, he had been company geologist for Malco Refineries, in Roswell.

Dennis K. Scott is in Cameo, W. Va., where he is general manager of the Cameo Coal Mining Co. He was with the Eastern Coal Corp., Stone, Ky.

Emil J. Servant, Jr., is maintenance superintendent for the Olga Coal Co. at Coalwood, W. Va.

Charles Will Wright returned just before Christmas from a ten-week trip to Greece, Italy, France, and Spain, and flew out to Denver to be with his son **Fred** over New Years. **Fred Wright,** who has been on mine methods research for the Bureau of Mines at the oil shale mine at Rifle, has recently had two major lung operations, but has recovered nicely.

John W. Vanderwilt

John W. Vanderwilt, well-known consulting geologist of Denver, will become president of the Colorado School of Mines on April, succeeding **Ben H. Parker.** The school's board of trustees accepted Dr. Parker's resignation with reluctance but realizes its good fortune in obtaining such a distinguished geologist, educator, and administrator as Dr. Vanderwilt to direct the school.

W. A. Wall has joined the staff of the Cobalt Chemical and Refinery Co., of Cobalt, Ont., as chief technical executive of the company's cobalt division. Mr. Wall formerly directed research in the leaching process for production of cobalt metal by the Rhokana Corp., and has also been connected with the departments of mines in Ontario and British Columbia.

George F. Zoffman is president of the Duval Texas Sulphur Co., producers of sulphur at Orchard, Texas, which has plans for a potash development program this year in the Permian Basin near Carlsbad. The company has core-drilled the area. **W. P. Morris** and **George Atwood,** formerly at the International Minerals and Chemical Corp. potash mine in the district, have joined the company.

Charles F. Williams

Charles F. Williams has moved his family to Santa Fe, N. Mex., from his cattle ranch in Kansas. He is associated with **Charles L. Bradbury,** of Albuquerque, in the ownership and operation of the old San Pedro Mines situated midway between the two cities. The San Pedro property was mined by the Spaniards before the American occupation and has a long record of production of gold, silver, copper, lead, and zinc ores.

Proposed Amendments to Bylaws

Notice is hereby given that at its meeting on Jan. 20, 1950, the Executive and Finance Committees voted to bring before the Board at its meeting next following that of

Feb. 13 the following changes in the Bylaws:

Substitute for the second and third sentences of Article 1, Section V, the following: "He shall be not more than thirty years of age at the time of his application, and shall not remain a Junior Member beyond his thirty-third birthday."

The effect of the amendment is to make the date of application rather than the date of election the limiting factor in the election of Junior Members, who may thus be permitted to attain their thirtieth birthday between the time of application and the time of election.

An amendment is also proposed to eliminate the distinction between Student Chapters and other Affiliated Student Societies, all of which would hereafter be known as Student Chapters. The proposed amendment is as follows:

In Article XI, Section 4 of the Bylaws, replace the words "Affiliated Student Society" wherever they occur by the words "Student Chapter," the words "Student Society" by "Student Chapter," and the word "Society" by "Chapter". Further, delete the last paragraph of this Section reading "An Affiliated Student Society whose bylaws require all members to be Student Associates of the Institute may use the designation "Student Chapter of AIME."

Obituaries

Frank Wyatt Collins (Member 1919), chief metallurgist with the Pan-American Engineering Co., died July 28, 1949. Mr. Collins was born at Dallas, Oreg., in 1883. From 1901 to 1905 he worked at the Union Iron Works in San Francisco during the day and did his technical studying at night. He was draftsman for several companies and in 1911 was made chief draftsman for the Chino Copper Co. For 2½ years he was with the Alaska Gastineau Mining Co. in Thane, returning to San Francisco to become chief engineer for Bradley Bruff & Labarthe. About twenty years ago he joined Pan-American Engineering. In 1948 he examined the Beavermouth operation on the Cottonwood River near Quesnel, B. C., for the Beavermouth Dredging Co.

Necrology

Date Elected	Name	Date of Death
1936	James E. Ash	Dec. 17, 1949
1922	Howard G. Bohlin	Jan. 25, 1950
1884	W. M. Claypool	Dec. 16, 1949
1919	Frank W. Collins	July 28, 1949
1891	Frank Drake	Jan. 14, 1950
1940	Harvey L. Garrity	Jan. 11, 1950
1928	William H. Hunt	Unknown
1934	W. A. Janssen	Dec. 17, 1949
1944	Thomas J. Newton	Dec. 19, 1949
1937	G. A. Peacock	Nov. 30, 1949
1906	Charles F. Potter	Dec. 1949
1918	William B. Price	Feb. 7, 1950
1913	Selden S. Rodgers	Jan. 8, 1950
1949	James J. Rogers, Jr.	Dec. 22, 1949
1948	F. N. Rupprecht	Dec. 29, 1949
1920	Edward R. Young	Jan. 9, 1949
1915	R. R. Van Valkenburgh	Oct. 29, 1949

George Samuel Rice

AN APPRECIATION BY
ARNO C. FIELDNER

George Samuel Rice, 83 (Member 1896; Honorary Member 1937), retired chief mining engineer of the Bureau of Mines, U. S. Department of the Interior, and internationally known authority on mine safety and mining research, died on Jan. 4, 1950, at Washington Sanitarium, Takoma Park, Md. Dr. Rice completed a long and useful life and lived to see the great saving in human lives that resulted from the pioneering explosion-prevention research he initiated in 1908. He has aptly been called the father of rock dusting in American coal mines. Tests in the Bureau's experimental coal mine at Bruceton, Pa., which he designed, first showed conclusively that bituminous coal dust and air will explode in a mine with no methane present and that rock dusting, thoroughly done, will render it inert.

Born in Claremont, N. H., in 1866, Dr. Rice received his mining engineering degree from Columbia University in 1887 and his honorary doctorate from Lafayette College in 1936. Following graduation from Columbia, he spent the next twenty

RIGHT FROM THE START
THIS PROPOSAL SAYS TO BUY TRUNION OVERFLOW MILLS, THEN, IF WE DON'T GET CAPACITY, ADD GRATES.
WHAT? THEN TEAR OUT FOUNDATIONS AND BUY ALL NEW MOTORS AND DRIVES ? I CAN SEE THE BOSS IF HE GOT A BILL FOR THAT!
OKAY, THEN LET'S GET SOME MORE MARCY GRATE DISCHARGE MILLS AND BE SURE WE'RE RIGHT.
YEAH, MARCYS HAVE ALWAYS GIVEN US LOWER POWER AND STEEL COSTS, WITH LOWER OVERALL GRINDING COSTS.
OTHER PRODUCTS
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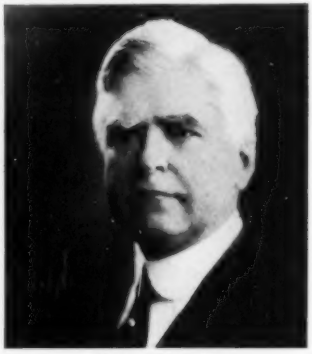
years in mining engineering, largely in Colorado, Pennsylvania, Iowa, and Illinois. His consulting work included considerable noncoal mining, such as development of new methods for mining pebble phosphate in Florida and the mining of potash minerals in the West.

An early paper before the Illinois Coal Mining Institute, discussing his own and foreign observations that fewer explosions occurred in high-ash coal mines, attracted the attention of Joseph A. Holmes, Director of the Technologic Branch of the U. S. Geological Survey. Dr. Holmes induced Dr. Rice to accept an appointment, in 1908, as chief mining engineer and take up a study of the cause of the disastrous coal mine explosions that caused the death of 1148 miners in eight mines in the United States in 1907.

Two years later, when mine-accident investigations were transferred to the newly established Bureau of Mines, Dr. Rice became chief mining engineer of the Bureau, with headquarters at Pittsburgh. Here he designed and constructed the first experimental coal mine in the world for studying the mechanism of coal dust explosions and demonstrated the efficacy of rock dusting in preventing propagation of explosions.

His work attracted the attention of foreign scientists, and in 1912 Dr. Rice organized an international conference on safety in mines research at Pittsburgh. Five other international conferences have been held since 1912, in England, France, Germany, Belgium, and the United States, the last one at Pittsburgh in 1948. As a representative of the Bureau of Mines, Dr. Rice was an active participant in these international conferences and, following the First World War, arranged for mutually valuable cooperation in research with the Safety in Mines Research Board of Great Britain which has continued to the present time.



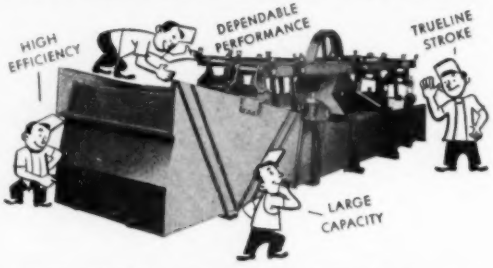
George S. Rice

During the Bureau's formative years in Pittsburgh, Dr. Rice, with the cooperation of James W. Paul and under the inspiring leadership of Director Holmes, assembled an enthusiastic group of young mining engineers to investigate mine explosions, develop rescue methods at disasters, and educate operators and miners in safer methods. Under his teaching, these men developed into the present-day leaders of safety and efficiency in coal mining as taught in our mining schools and as practiced in industry.

As the work of the Bureau of Mines expanded, the office of the chief mining engineer was transferred to Washington in 1916, although Dr. Rice continued to direct the Experimental mine explosion investigations until his retirement from the Bureau.

On the entry of the United States into World War I, Dr. Rice suggested that the Bureau of Mines, through its chemists and engineers experienced in dealing with explosives and poisonous gases in mines, would be of great assistance in developing masks for protecting our troops against war gases. This project was instituted by the Bureau and later formed the nucleus of

MORSE *Trueline* RAKE CLASSIFIERS



Dependable classifiers that improve performance, simplify operation and lower maintenance.

Large and quiescent pool areas give high overflow efficiency. Long "Trueline" strokes, parallel to the tank bottom, discharge the maximum quantity of sands. Fabricated steel construction gives extra strength and long service.

COMPACT DRIVE—SMOOTH OPERATION—METALLURGICALLY RIGHT

MB **MORSE BROS. MACHINERY COMPANY** Established 1898

WRITE FOR BULLETIN

Denver, Colo. U.S.A.

the newly organized Chemical Warfare Service of the Army. At the close of the war, Dr. Rice reported on war damage to French and Belgian mines and metallurgical plants.

Another example of Dr. Rice's vision in solving pioneering problems by research was his suggestion of a cooperative investigation by the Bureau of Mines for determining the ventilation requirements of the Holland vehicular tunnels under the Hudson River between New York and New Jersey.

In the two decades following the War, Dr. Rice kept in close touch with new European developments by visits to foreign countries and as the Bureau of Mines representative at international conferences. The published results of these studies included reports on safety in mines, methods of mining oil shale in Scotland, mining oil sand and potash minerals in France and Germany, hydraulic stowing to prevent or minimize surface subsidence, outbursts of gas in coal mines, and a comprehensive survey of coal mining in Europe.

Dr. Rice was the author of numerous technical papers and bulletins on safety and mining investigations.

In recognition of his services in behalf of mine safety, Dr. Rice was awarded medals by the British Institution of Mining Engineers, of which he was made an honorary member, in 1923, by his alma mater, Columbia University, in 1931, and by the Joseph A. Holmes Safety Association in 1946.

Membership

Total AIME membership on Jan. 31, 1950, was 16,441; in addition 5010 Student Associates were enrolled.

ADMISSIONS COMMITTEE

less, T. B. Counselman, Ivan A. Given, Phillips, Vice-Chairman; George B. Cor- James L. Head, Chairman; Albert J. Robert L. Hallett, Richard D. Mollison, and John Sherman.

Institute members are urged to review this list as soon as the issue is received and immediately to wire the Secretary's Office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by air mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; AM, Associate Member; S, Student Associate.

Alabama

Lewisburg—Sides, Judson B. (M).
Russelville—Britton, Sam Abney. (R,C/S—JA-M).

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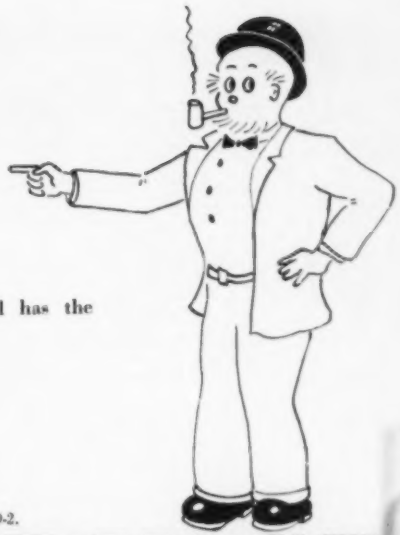

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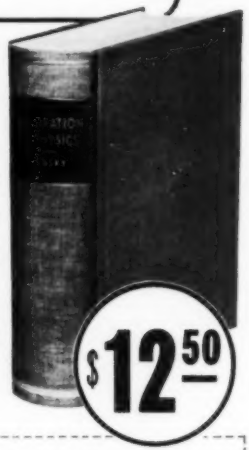

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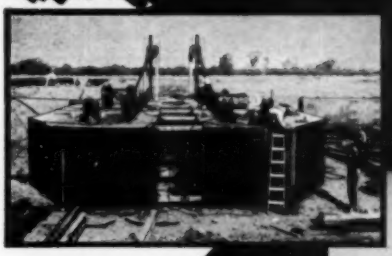

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